TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XXVIII.

FEBRUARY, 1898, TO OCTOBER, 1898.

INCLUSIVE.

NEW YORK CITY:

PUBLISHED BY THE INSTITUTE,

AT THE OFFICE OF THE SECRETARY.

1899.

PREFACE.

THE preparation of this volume has been somewhat delayed, and the attendant labor greatly enhanced, by the illness and death of my dear friend and accomplished editorial assistant, Mrs. Helen Stevens Conant, who died April 17, 1899. Mrs. Conant was thoroughly trained in literary work, mistress of several foreign languages, familiar with all the details of publishing from authorship to proof-reading, and able to maintain at all times a complete knowledge and control of the complicated business of that department of the Secretary's work. The mere statement that this department has on hand, at any given period, an average of more than sixty papers, in various stages of publication, from the original manuscript to the final sheets (*nine times* revised) of the volume; that proofs of text and engravings are sent repeatedly to authors in all parts of the civilized world, as well as to special revisers outside of this office, besides being carefully examined here; that it is necessary to know at any moment the exact stage of every separate publication, and to secure by persistent vigilance the final preparation of all in time for the publication of the annual volume of *Transactions;* and that this volume must be ultimately examined for Errata, and indexed with fullness and accuracy—this outline of the work, I say, will sufficiently indicate the continuous, varied and onerous labor which it involves.

No one but myself will ever know how much of the credit universally given to our *Transactions* for exceptional freedom from errors of the pen or the press was due to this patient and skillful worker at my side.

R. W. RAYMOND.

NEW YORK, June, 1899.

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OFFICERS.

For year ending February, 1899.*

President.

CHARLES KIRCHHOFF, New York City.

Vice-Presidents

D. W. BRUNTON,	Aspen, Colo.
W. E. C. EUSTIS,	Boston, Mass.
JAMES DOUGLAS,	New York City.

(Term expires February, 1899.)

E. D. PETERS, JR.,	Dorchester, Mass.
ALBERT R. LEDOUX,	New York City.
LEON P. FEUSTMAN,	Mexico City, Mexico.

(Term expires February, 1900.)

Managers.

JAMES GAYLEY,	Pittsburgh, Pa.
JAMES F. KEMP,	New York City.
BENJ. SMITH LYMAN,	Philadelphia, Pa.

(Term expires February, 1899.)

C. W. GOODALE,	Butte, Mont.
FRANK LYMAN,	Brooklyn, N. Y.
FRANK McM. STANTON,	Houghton, Mich.

(Term expires February, 1900.)

R. P. ROTHWELL,	New York City.
W. J. OLCOTT,	Duluth, Minn.
W. B. DEVEREUX,	Glenwood Springs, Colo.

(Term expires February, 1901.)

Secretary.

R. W. RAYMOND,
99 John St, New York City.

Treasurer.

THEODORE D. RAND,
Philadelphia, Pa.

Honorary Members.

PROF. RICHARD ÅKERMAN, Stockholm, Sweden; SIR LOWTHIAN BELL, Middlesborough, England; DR. THOMAS M. DROWN, South Bethlehem, Pa.; PROF. HATON DE LA GOUPILLIÈRE, Paris, France; PROF. HANS HOEFER, Leoben, Austria; DR. BRUNO KERL, Berlin, Germany; PROF. JOSEPH LE CONTE, Berkeley, Cal.; PROF. J. P. LESLEY, Philadelphia, Pa.; PROF. THEODOR RICHTER,† Freiberg, Germany; BERGRATH ALBERT SERLO,† Berlin, Germany; PROF. DR. HERMANN WEDDING, Berlin, Germany.

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* The following officers were elected at the Annual Meeting, February, 1899: *President*, James Douglas, New York City; *Vice-Presidents* (to serve two years), E. C. Potter, Chicago, Ill.; George F. Kunz, New York City; W. N. Page, Ansted, W. Va.; *Managers* (to serve three years), Arthur Winslow, St. Louis, Mo.; William Glenn, Baltimore, Md; W. J. Taylor, Bound Brook, N. J.; *Treasurer*, Theodore D. Rand, Philadelphia, Pa.; *Secretary*, Rossiter W. Raymond, New York City.

† Died during 1898.

PROF. SIR WILLIAM C. ROBERTS-AUSTEN, London, Eng., and M. FLORIS OSMOND, Paris, France, were elected honorary members at the Annual Meeting, February, 1899.

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* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting, May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

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* Annual meeting for the election of officers.
† Begun in February at New York City, for the election of officers, and adjourned to Florida.

PUBLICATIONS.

THE publications of the Institute comprise:

PAMPHLETS.

1. The minutes of the Proceedings of each Meeting.

2. Such of the papers presented or read by title at each Meeting as are furnished by the authors and approved by the Council for full publication. (In nearly all cases in which papers, the titles of which appear in the Proceedings, are not subsequently published, they have been withdrawn by the authors.) These papers are published separately in pamphlet form, and are marked "subject to revision." Beyond the edition distributed without charge to members and associates not in arrears, a small supply is retained to meet subsequent demand. There are no copies on hand of papers read before 1880. The stock is nearly complete from 1880. These papers are for sale at the office of the Secretary, or are sent to purchasers by mail or express, charges paid, on receipt of the price, as follows:

NO. OF PAGES	SINGLE COPIES.	10 COPIES.	20 COPIES.
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21 to 24 "	0 20	1 50	2 75
25 to 40 "	0 25	2 00	3 50
41 to 56 "	0 30	2 50	4 50
57 to 72 "	0 35	3 00	5 00
73 to 88 "	0 40	3 25	5 25
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TRANSACTIONS.

The volumes of *Transactions*, which are published annually, contain the list of officers, rules, etc., the Proceedings, and the papers revised for final publication. (In this revision, after the preliminary publication, authors are permitted to use the largest liberty; and the changes and additions made in papers are sometimes important. It should be borne in mind, by those who study or quote a paper in the preliminary edition, that they may not have in that form the ultimate and deliberate expression of the author's views. It should be added, however, that in the majority of cases there is no important change.) These volumes are for sale as follows, in paper covers:

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Extra copies, when ordered before the types have been distributed, are furnished to authors, under Rule VII., at the following rates:

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17 to 20 "	3 25	4 50	6 25	1 50
21 to 24 "	3 75	5 25	7 25	1 75
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Covers (including printing on first page of the same), extra	1 50	2 00	3 00	0 50

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All communications and remittances should be addressed to R. W. Raymond, Secretary, P. O. Box 223, New York City.

RULES

ADOPTED MAY, 1873. AMENDED MAY, 1875, 1877, AND 1878, FEBRUARY, 1880, 1881, 1887, 1890, AND 1896.

I.

OBJECTS.

THE objects of the AMERICAN INSTITUTE OF MINING ENGINEERS are to promote the arts and sciences connected with the economical production of the useful minerals and metals, and the welfare of those employed in these industries, by means of meetings for social intercourse, and the reading and discussion of professional papers, and to circulate, by means of publications among its members and associates, the information thus obtained.

II.

MEMBERSHIP.

The Institute shall consist of Members, Honorary Members, and Associates. Members and Honorary Members shall be professional mining engineers, geologists, metallurgists, or chemists, or persons practically engaged in mining, metallurgy, or metallurgical engineering. Associates shall include all suitable persons desirous of being connected with the Institute, and duly elected as hereinafter provided. Each person desirous of becoming a member or associate shall be proposed by at least three members or associates, approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe) upon receiving three-fourths of the votes cast, and shall become a member or associate on the payment of his first dues. Each person proposed as an honorary member shall be recommended by at least ten members or associates, approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe) on receiving nine-tenths of the votes cast; *Provided*, that the number of honorary members shall not exceed twenty. The Council may at any time change the classification of a person elected as associate, so as to make him a member, or *vice versa*, subject to the approval of the Institute. All members and associates shall be equally entitled to the privileges of membership; *Provided*, that honorary members shall not be entitled to vote, and members or associates whose post-office address shall be outside of the United States, Canada and Mexico shall not be entitled to vote by mail, except upon proposed amendments to the Rules.

Any member or associate may be stricken from the list on recommendation of the Council, by the vote of three-fourths of the members and associates present at any annual meeting, due notice having been mailed in writing by the Secretary to the said member or associate.

III.

DUES.

The dues of members and associates shall be ten dollars, payable upon their election, and ten dollars per annum thereafter, payable in advance on the first day of each calendar year. Honorary members shall not be liable to dues. Any member or associate not in arrears may become by the payment of one hundred dollars at one time a life-member or associate, and shall not be liable thereafter to annual dues. Any member or associate in arrears may, at the discretion of the Council, be deprived of the receipt of publications, or stricken from the list of members when in arrears for one year; *Provided*, that he may be restored to membership by the Council on payment of all arrears, or by re-election after an interval of three years.

IV.

OFFICERS.

The affairs of the Institute shall be managed by a Council, consisting of a President, six Vice-Presidents, nine Managers, a Secretary and a Treasurer, who shall be elected from among the members and associates of the Institute at the annual meetings, to hold office as follows:

The President, the Secretary, and the Treasurer for one year (and no person shall be eligible for immediate re-election as President who shall have held that office subsequent to the adoption of these rules, for two consecutive years), the Vice-Presidents for two years, and the Managers for three years; and no Vice-President or Manager shall be eligible for immediate re-élection to the same office at the expiration of the term for which he was elected. At each annual meeting a President, three Vice-Presidents, three Managers, a Secretary, and a Treasurer shall be elected, and the term of office shall continue until the adjournment of the meeting at which their successors are elected.

The duties of all officers shall be such as usually pertain to their offices, or may be delegated to them by the Council or the Institute; and the Council may in its discretion require bonds to be given by the Treasurer. At each annual meeting the Council shall make a report of proceedings to the Institute, together with a financial statement.

Vacancies in the Council may occur by death or resignation; or the Council may, by a vote of the majority of all its members, declare the place of any officer vacant, on his failure for one year, from inability or otherwise, to attend the Council meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; *Provided*, that the said appointment shall not render him ineligible at the next annual meeting.

Five members of the Council shall constitute a quorum; but the Council may appoint an Executive Committee, or business may he transacted at a regularly called meeting of the Council, at which less than a quorum is present, subject to

the approval of a majority of the Council, subsequently given in writing to the Secretary, and recorded by him with the minutes.

V.

ELECTIONS.

The annual election shall be conducted as follows: Nominations may be sent in writing to the Secretary, accompanied with the names of the proposers, at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before the said meeting, mail to every member or associate (except honorary members) a list of all the nominations for each office so received, together with a copy of this rule, and the names of the persons ineligible for election to each office; and if the Council, or a Committee thereof, appointed for the purpose, shall have recommended any nominations, such recommendation may also be sent to members and associates with the said list of all nominations made, but not upon the same paper. And each member or associate, qualified to vote, may vote, either by striking from or adding to the names of the said list, leaving names not exceeding in number the officers to be elected, or by preparing a new list, signing said altered or prepared ballot with his name, and either mailing it to the Secretary or presenting it in person at the annual meeting; *Provided*, that no member or associate in arrears since the last annual meeting shall be allowed to vote until the said arrears shall have been paid. The ballots shall be received and examined by three Scrutineers, appointed at the annual meeting by the presiding officer; and the persons who shall have received the greatest number of votes for the several offices shall be declared elected, and the Scrutineers shall so report to the presiding officer. The ballots shall be destroyed, and a list of the elected officers, certified by the Scrutineers, shall be preserved by the Secretary.

VI.

MEETINGS.

The annual meeting of the Institute shall take place on the third Tuesday of February, at which a report of the proceedings of the Institute and an abstract of the accounts shall be furnished by the Council. Other meetings shall be held in each year, at such times and places as the Council shall select, and notice of all meetings shall be given by mail, or otherwise, to all members and associates, at least twenty days in advance.

Every question which shall come before any meeting of the Institute, shall be decided, unless otherwise provided by these Rules, by the votes of a majority of the members then present. Any member or associate may introduce a stranger to any meeting; but the latter shall not take part in the proceedings without the consent of the meeting.

VII.

PAPERS AND PUBLICATIONS.

The Council shall have power to decide on the propriety of communicating to the Institute any papers which may be received, and they shall be at liberty, when they think it desirable, to direct that any paper read before the Institute shall

be printed in the *Transactions*. Intimation, when practical, shall be given, at each general meeting, of the subject of the paper or papers to be read, and of the questions for discussion at the next meeting. The reading of papers shall not be delayed beyond such hour as the presiding officer shall think proper; and the election of members or other business may be adjourned by the presiding officer, to permit the reading and discussion of papers. The published papers and volumes of *Transactions* shall be distributed to all members and associates not in arrears, and may be sold to the public upon such conditions as the Council shall prescribe; but the Council may, in its discretion, omit sending to members and associates outside of the United States, Canada and Mexico, special circulars, unless the same contain proposed amendments to the Rules.

The copyright of all papers communicated to, and accepted by, the Institute, shall be vested in it, unless otherwise agreed between the Council and the author. The author of each paper read before the Institute shall be entitled to twelve copies, if printed, for his own use, and shall have the right to order any number of copies at the cost of paper and printing, provided said copies are not intended for sale. The Institute is not, as a body, responsible for the statements of fact or opinion advanced in papers or discussions at its meetings, and it is understood that papers and discussions should not include matters relating to politics or purely to trade; nor shall the Council or the Institute officially approve or disapprove any technical or scientific opinion or any proposed enterprise outside the management of the meetings, discussions and publications of the Institute, as provided in these Rules; *Provided*, however, that committees may be appointed by the Council or the Institute to make investigations and submit reports at meetings of the Institute; but no action shall be taken binding the Institute for or against the conclusions of any such reports.

VIII.

AMENDMENTS.

These Rules may be amended at any annual meeting by a two-thirds vote of the members present; *Provided*, that written notice of the proposed amendment shall have been given at a previous meeting; *and Provided, also*, that the amendment or amendments so adopted shall be printed upon a ballot and sent, not later than the next distribution of printed matter, to all members and associates not in arrears for the preceding year (except honorary members and foreign members elected before February, 1880), and each person receiving the same shall be requested to return it to the Secretary with his written vote of Yes or No to each amendment, and his signature; and the President shall appoint as Scrutineers three members or associates, who shall examine all of the said ballots which shall have been returned within one month from the date of their distribution, and shall report the result; and the Secretary shall publish and distribute to members, not later than the next distribution of printed matter, an announcement of the said result so reported, together with the text of the additional or amended rule or rules so adopted; and the amendment or amendments approved by the majority of the ballots so returned and reported shall become part of these Rules from and after the publication of said announcement by the Secretary.

Proceedings of the Seventy-Fourth (Twenty-Eighth Annual) Meeting, Atlantic City, N. J., February, 1898.

COMMITTEES.

THERE being no members of the Institute resident in Atlantic City, no Local Committee of Reception was formed, and the arrangements for the meeting, sessions, etc., excepting the proceedings of Friday, as specified below, were left to the Secretary, and were efficiently carried out, under his general direction, by Mr. Theodore Dwight, Assistant Treasurer of the Institute.

The entertainment of the Institute in Philadelphia on Friday was provided by the following Philadelphia members:

PHILADELPHIA COMMITTEE.

John Birkinbine, *Chairman;* Charles P. Bower, William Burnham, Dr. H. M. Chance, Walton Clark, John H. Converse, James M. Dodge, H. S. Drinker, Theodore N. Ely, J. L Ferguson, Stanley G. Flagg, Jr., Dr. Persifor Frazer, F. L. Garrison, John M. Hartman, Washington Jones, E. K. Landis, W. K. Mitchell, W. H Morris, W. G. Neilson, Dr. R. A. F. Penrose, Theodore D. Rand, Dr. Charles Schaffer and Jones Wister.

Hotel headquarters were established at Haddon Hall, and sessions were held at the Casino, belonging to the proprietors of the Hotel Brighton, and by their courtesy specially opened, warmed and lighted, and placed at the disposal of the Institute.

The opening session was held on Tuesday evening, February 15th, in the Casino. After brief introductory remarks by President Drown, a paper was read by Mr. Gifford Pinchot, New York City, on Mining and the Forest Reserves, illustrated by numerous lantern-views, showing the injurious results of neglect, waste and forest-fires upon the timber-reserves available to the mining industry.

A paper was then read by Mr. S. F. Emmons, Washington, D. C., on A Geological Journey through Southern Russia, illustrated by a most interesting series of lantern-views, presenting the scenery, topography, inhabitants, etc., of the regions described.

The President announced the appointment of Messrs. B. F. Fackenthal, Jr., J. H. Lee and E. K. Landis as the Scrutineers appointed to examine the ballots received for officers, and report the result at a later session.

The second session was held at the Casino on Wednesday afternoon, at 3 o'clock. After announcements of arrangements, etc., and the election of a large number of new members (see list below), Dr. Persifor Frazer, by request of the President, made a brief statement, calling attention to a number of curious and interesting objects from Russia, exhibited in connection with his lecture (delivered at a later session), and comprising specimens of painting upon wood, lace-work from Southern Russia, gem-cutting from the celebrated lapidaries of Ekatérineburg, etc. The article of greatest metallurgical interest was a cast-iron medal, in honor of the geologists of the Ural excursion of the VIIth International Geological Congress. This medal was cast at Kytchtym, an important plant, situated on the Asiatic slope of the Ural range, midway between Tschéliabinsk and Ekatérineburg, on the arm of the Siberian railway, which turns abruptly north at the former town, and runs for about 100 miles parallel with the range to the latter. The specimen exhibited by Dr. Frazer had never been touched with tool or brush, polished or finished in any way, and retained the ribbon-like sprue of metal from the runner, through which the unusually fluid metal had been run into the mould. The delicacy and sharpness of the casting were exceptional, as was also its freedom from shrinkage. The German and English iron-experts with the party agreed that no such work had ever been accomplished in their respective countries.

The following papers were read and discussed:

An Automatic Feed-Device for Gas-Producers, by C. W. Bildt, Worcester, Mass.

An Apparatus for Removing the Sand from the Waste-Water of Ore-Washers, by J. E. Johnson, Jr., Longdale, Va.

Note on the Use of the Tri-axial Diagram and Triangular Pyramid for Graphical Illustration, by H. M. Howe, New York City.

The following papers were read by the Secretary, in the absence of the authors, and discussed:

Notes on the Bertrand-Thiel Process, by Joseph Hartshorne, Stowe, Pa.

Notes on the Stockholm Exposition and the Iron and Steel Trade of Sweden, by James Douglas, New York City.

The Secretary presented continued discussion of the Physics of Cast-Iron, received by mail.*

The third session was held at the Casino on Wednesday evening, February 16th. Before the commencement of the regular session Mr. Theodore Dwight, New York City, exhibited, by request, a number of exquisitely beautiful and technically wonderful lantern-views, from prize slides, courteously lent for the occasion by the New York Camera Club; also, a number of Mexican views, interesting to those who are intending to take part in a projected meeting of the Institute in Mexico.

Dr. Persifor Frazer, with a few appropriate words, offered the following resolutions, which were unanimously adopted:

"*Resolved*, That the American Institute of Mining Engineers, at its annual meeting in Atlantic City, has heard with intense horror of the national calamity which has occurred in the harbor of Havana by the destruction of the battle-ship Maine.

"*Resolved*, That the American Institute of Mining Engineers tenders its sincere sympathies to the families of the officers and sailors who lost their lives in this appalling catastrophe."

The following papers were then presented:

The New Breaker at Cranberry Coal-Mine, Hazleton, Pa., by W. S. Ayres, Hazleton, Pa. (Illustrated with lantern-views.)

Notes on the Geological Structure of the Caucasus Range along the Georgian Military Road, by Dr. Persifor Frazer, Philadelphia, Pa.†

The fourth and concluding session was held at the Casino on Thursday morning, February 17th. Additional members were elected (see list below).

The following papers were presented by the Secretary, in the absence of the authors, and discussed:

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* Published in vol. xxvii. See Preface to that volume.

† In connection with this subject, Dr. Frazer delivered an interesting lecture, illustrated with numerous views taken during the journeys connected with the recent VIIth International Geological Congress, and comprising much more than the paper printed in this volume.

A New Form of Ingot-Mould for Casting Brass or Bronze Ingots, with Remarks on the General Form of Ingots, by Erwin S. Sperry, Bridgeport, Conn.

The Influence of Antimony on the Cold-Shortness of Brass, by Erwin S. Sperry, Bridgeport, Conn.

A Study of the Elimination of Impurities from Copper-Mattes in the Reverberatory and Converter, by Edward Keller, Baltimore, Md.

The Relation of the Strength of Wood under Compression to the Transverse Strength, by B. E. Fernow, Washington, D. C.

Discussion received by mail of Dr. Don's paper (Chicago meeting, February, 1897) on the Genesis of Certain Auriferous Lodes,* and of Prof. Kidwell's paper (Lake Superior meeting, July, 1897) on the Efficiency of Built-Up Wooden Beams was read by the Secretary, and oral discussion of the latter paper followed.*

The following paper was read and discussed:

Note on Limonite Pseudomorphs from Dutch Guiana, by R. W. Raymond, New York City.

Discussion received by mail of the paper of Mr. Heath (Lake Superior meeting, July, 1897) on the Electrolytic Assay as Applied to Refined Copper, was presented by the Secretary.*

The following papers were presented in printed form:

The Kotchkar Gold Mines, Ural Mountains, Russia, by H. B. C. Nitze, Baltimore, Md., and C. W. Purington, Boston, Mass.

The Ultimate and the Rational Analysis of Clays and their Relative Advantages, by Heinrich Ries, New York City.

Emery, Chrome-Ore and other Minerals in the Villayet of Aidin, Asia Minor, by W. F. A. Thomae, London, England.

Mining Districts of Colombia, S. A., by Henry C. Granger and Edward B. Treville, Quibdo, Colombia, S. A.

Kalgoorlie, Western Australia, and its Surroundings, by George J. Bancroft, Denver, Colo.

The following papers were read by title, to be subsequently printed and distributed to members:

The Manganese-Ore Industry of the Caucasus, by Frank Drake, New York City.

Some Peculiar Occurrences of Gold-Ore in Montana, by W. H. Weed, Washington, D. C.

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* Published in vol. xxvii. See Preface to that volume.

Tennessee Phosphates, by Lucius P. Brown, Nashville, Tenn.

Sectional Cushioned Rolls, by J. W. Pinder, Elko, Nev.

Scorification and Cupellation without Muffle.—A New Furnace and Method for Gold and Silver Assays, by George A. Koenig, Houghton, Mich.

Notes of a Reconnaissance from Springfield, Mo., into Arkansas, by E. J. Schmitz, New York City.

Stamp-Mill Indicator-Diagrams, by Henry Louis, Newcastle-upon-Tyne, England.

The Auriferous Deposits of Siberia, by René de Batz, Paris, France.

Notes on the Vein-Formation and Mining of Gilpin County, Colo., by Forbes Rickard, Central City, Colo.

The Scrutineers reported the following persons to have been elected as officers of the Institute:

PRESIDENT.

CHARLES KIRCHHOFF, ...	New York City.

VICE-PRESIDENTS.

(To serve two years.)

E. D. PETERS, JR., ...	Dorchester, Mass.
A. R. LEDOUX, ...	New York City.
LEON P. FEUSTMAN, ...	Mexico City, Mexico.

MANAGERS.

(To serve three years.)

R. P. ROTHWELL, ...	New York City.
W. J. OLCOTT, ...	Duluth, Minn.
W. B. DEVEREUX, ...	Glenwood Springs, Colo.

TREASURER.

THEODORE D. RAND, ...	Philadelphia, Pa.

SECRETARY.

ROSSITER W. RAYMOND, ...	New York City.

The Annual Report of the Council was presented as follows:

ANNUAL REPORT OF THE COUNCIL.

In accordance with the rules, the Council makes the following report to the Institute:

The financial statement of the Secretary and Treasurer shows receipts from all sources for the year ending December 31st (in-

cluding $8721.76 on hand at the beginning of the year, and $4598 withdrawn from special deposit), of $42,189.10, and expenditures (including $11,992.50 for purchase of securities, as explained below) of $39,223.45, leaving a cash balance of $2965.65, a reduction in cash on hand, as compared with the balance of last year, of $5756.11. This is explained by the investments alluded to, as the result of which the Treasurer now holds, besides United States bonds to the par value of $2900, general mortgage 6 per cent. bonds of the Pennsylvania Railroad Company, of the par value of $9000, which cost $11,992.50. The total securities thus held represent the funded payments of life members. Since these investments were made, additional payments for life-membership have raised the number of such members to 178; and further investments will be made, from time to time, to cover the net increase of this list.

Apart from these features, it will be seen that the actual receipts of the year (including $1962.34 for life-memberships*) were $28,869.34, and the expenses (including $1218.32 for preparing, printing and binding the extra Index Volume, besides a further sum, not separated in the accounts, but estimated to amount to about $200, for the distribution of that volume) were $27,230.95, showing an excess of receipts over expenses of $1638.39, or somewhat less than the payments for life-membership. In other words, all the receipts for the year have been practically expended for the benefit of the members, which is, in the opinion of the Council, the best disposition that could be made of them.

The detailed statement of receipts and disbursements is as follows:

Receipts.

Balance from statement,		$8,721.76
Withdrawn from special deposit,		4,598.00
Annual dues,	$22,555.88	
Life membership,	1,962.34	
Binding of *Transactions*,	1,861.19	
Sale of publications,	1,485.21	
Electrotypes,	69.00	
Interest on bonds and deposits,	935.72	28,869.34
		$42,189.10

* The odd figures are due to variations in exchange, affecting the remittances of foreign members.

Disbursements.

Extra expense of preparing and proof-reading of Index XXI. to XXV.,	$300.00	
Printing Volume XXVI. of *Transactions*,	3,156.37	
" Index XXI. to XXV.,	508.14	
" pamphlet edition of papers,	2,710.22	
" mailing list,	37.00	
" circulars and ballots,	215.81	
Binding Volume XXVI. and miscellaneous Volumes of *Transactions*,	1,817.05	
Binding Index XXI. to XXV.,	410.18	
" exchanges,	46.18	
Engraving and electrotyping,	1,365.40	
Secretary's department, including clerks, stenographers, and expenses of editing and proof-reading,	7,991.00	
Postage, including Post-Office box rent,	836.85	
Stationery,	217.52	
Rent,	800.00	
Express and freight charges,	2,191.93	
Telephone,	121.70	
Telegrams, cablegrams and car-fare,	40.95	
Coal, ice and gas,	66.15	
Assistant Treasurer's department,	2,853.57	
Storage of *Transactions*,	103.68	
Special stenographers and expenses of meetings,	931.05	
Office supplies and repairs,	443.78	
Insurance,	58.25	
Miscellaneous,	8.17	
	$27,230.95	
Purchase $9000 Pennsylvania General Mortgage, 6 per cent.,	11,992.50	
	$39,223.45	
Balance,	2,965.65	
	$42,189.10	$42,189.10

In addition to the usual publication and distribution of papers in pamphlet form and of the Annual Volume (Volume XXVI., comprising 1141 pages) the Institute has issued and distributed, during the past year, without extra charge to members, a bound volume, containing a complete analytical index of Volumes XXI. to XXV. inclusive. Taken together with previous similar index-volumes, this furnishes a complete detailed guide to the contents of the *Transactions* for a quarter of a century, and confers upon the volumes the value of a cyclopædia, covering the progress of mining and metallurgy for the most eventful and important period in the history of those arts.

The Institute still possesses a few complete sets of the *Transactions;* and the Council, deeming them to be assets, the value of which will increase with time, is not anxious to force the sale of them; but, in the interest of members, attention is called to the advisability of completing their sets, while this is still practicable, by the purchase of back-volumes. At present, such volumes may be had singly; but as the supply approaches exhaustion, it is possible that the Council may decide to sell complete sets only, or to increase the price for single volumes.

Two meetings were held during the year, the annual meeting at Chicago in February, and a Lake Superior meeting in July. The printed "Proceedings" of these meetings, already received by members, have amply proved their professional interest and social success; and Volume XXVII. of the *Transactions*, now in preparation, will show that their contributions to technical literature were not inferior in value to those of preceding years. In this connection, it may be well to emphasize the fact that, by reason of the diversified character and widely-scattered distribution of the membership of the Institute, the permanent value of the results of one of its meetings, to those not attending it, cannot be measured by the actual proceedings at the sessions. The real public addressed by the author of an Institute paper is not the company which happens to be present at the meeting at which it is presented, but the wider constituency which receives it in print. Judged by this standard, the papers of the past year will be found to be highly important and interesting, though some of the most notable among them have been simply "read by title" at the meeting.

Changes in membership have taken place during the year as follows: 209 members and 32 associates have been elected; 10 associates have become members; the deaths of 1 honorary member, 17 members and 1 associate have been reported; 65 members and 5 associates have resigned; and 65 members and 6 associates have been dropped for continued default in the payment of dues. These changes are tabulated as follows, showing a net gain of 81 in total membership, which will be largely augmented by the list of candidates for membership to be elected at the present meeting:

	H. M.	F. M.	M.	A.	Totals.
At date of last report	12	38	2203	164	2417
Gains: By Election	....	....	209	32	241
Change of Status	....	....	10	....	10
Losses: By Resignation	....	....	65	5	70
Dropping	....	....	65	6	71
Change of Status	....	....	....	10	10
Death	1	1	16	1	19
Total gains	....	....	219	32	251
Total losses	1	1	146	22	170
Present membership	11	37	2276	174	2498

The list of deaths comprises the names of Peter Ritter von Tunner, honorary member, A. Noblet, foreign member (of the class existing until 1880), and the following members and associates: David A. Bennett (1889), Rudolph Boericke (1893), E. Borda (1875), P. W. Duffield (1889), George W. Goetz (1882), John A. Grant (1896), Joseph Hunt (1871), Alexis Janin (1885), A. Leon Jousselin (1888), Walter Marsh (1888), J. H. R. Molson (1889), Thomas R. Morgan (1878), W. S. Nelson (1895), William Rotthoff (1895), George A. Spotswood (1889), John Thomas (1871) and F. A. Thompson (1894).

Of these, the names of Ritter von Tunner and George W. Goetz have been suitably commemorated by special Biographical Notices in the *Transactions*. The following data, comprising what has been received by the Secretary as to other names in the list, is here placed on record. Any appropriate biographical data concerning names not here mentioned, which may be received hereafter, will be included in the next annual report.

Rudolph Boericke was born in 1864, and died on Christmas day, 1897, from exhaustion, the result of exposure and exertion, connected with the capsizing of a boat, in which he had been paddling with his brother. He was a graduate of the University of Pennsylvania, and, before going to college, had served an apprenticeship at Cramp's ship-yard, near Philadelphia. Subsequently, he was assistant engineer on one of the ships of the United States and Brazil Mail Steamship Company, and, for a time, in the employ of the Westinghouse Electric Company. During the latter years of his brief but active professional life he made a careful study of fuels and their treatment, and associated himself with Mr. W. M. Stein, of Philadelphia,

in this branch of business. Shortly before his death he had returned from Texas, where he had spent a year in the erection for his firm of a large briquette-plant. The lamentable accident which cost him his life cut short a fruitful and promising career.

Eugene Borda, for about forty years a leading Philadelphia dealer and operator in coal, and for twenty-two years a member of the Institute, was born in 1825 at Paris, France, and died February 8, 1897 at Philadelphia. At the age of 25, he came to this country, and superintended for a short time the Chestnut Hill ore-bank, Pa., and subsequently for 10 years the mining operations of his uncle, Mr. Charles A. Heckscher, at Woodside, near Minersville, Pa., where he acquired a thorough knowledge of the coal business. In 1862 he removed to Philadelphia, and engaged in the business of shipping and dealing in coal, which he continued, first as a partner with others, then on his own account, and finally (1884), in partnership with his son, until the time of his death.

Patricio Wilson Duffield was born in 1856 at Bogota, Colombia, S. A., and educated in England, studying for a time at the Royal School of Mines, London. He came subsequently to Canada and the United States, where he was practically engaged in mining and milling until 1881, when he settled in New York City as a member of the engineering firm of McDermott & Duffield. In 1887 he removed to London. After a visit to Africa, he passed several years in Mexico, where, besides practicing as a mining engineer, he represented the firm of Fraser & Chalmers, of Chicago. In 1895 he went to Western Australia as engineer for the Exploration Company of London. While there, he contracted typhoid fever, and, still later, he suffered in New Zealand an attack of illness from which he never fully recovered. He died in London, July 7, 1897. Mr. Duffield was personally known and highly esteemed by many members of the Institute, who appreciated not only his professional ability and energy, but also his conspicuously happy disposition and indomitable cheerfulness.

Joseph Hunt, one of the earliest and most estimable members of the Institute, was born in 1844 at Philadelphia, Pa. He was of English ancestry, his great-grandfather having come to America when the Penn colony was established. His father,

Thomas Hunt, was a pioneer in iron rolling-mills, both building and superintending them. In his youth, Mr. Hunt learned the trade of machinist in works at Wilmington, Del. During the war of the rebellion, he was an assistant engineer in the United States Navy. After the war, he studied civil and mining engineering. About 1867 he superintended iron furnaces at Richmond, Va.; subsequently he built the Bingen furnace in Northampton county, Pa., and still later he entered the employ of the Crane Iron Company, Pa., of which he afterwards became, and remained for 15 years, the superintendent. During his superintendency the works of the Company were practically rebuilt and modernized. In 1887 he resigned to accept another position, in which he built furnaces at Emporium, Pa. At a later period he took charge of the Allentown, Pa., Foundry and Machine Company; and finally, about one year before his death, he was appointed, after passing brilliantly the requisite examination, a government inspector of armor, etc., in the works at South Bethlehem, Pa. He was actively engaged in this work when he suddenly died of heart-disease.

Alexis Janin belonged to a family distinguished in the history of American mining and metallurgy. His ability in practice was well known on the Pacific coast; but he made no important contributions to the public literature of his profession, and, so far as is now known to his colleagues, the great and varied fruits of his experience died with him.

Walter Marsh studied from 1876 to 1879 at the Royal School of Mines, London, and after several years of practice in England went to India, where he was employed by the government in mining explorations. Subsequently he was for four years (1887–1891) manager of the Iron-Clad Gold Mines, Cargo, New South Wales, and subsequently for a year general manager of the British Broken Hill Proprietary Company. Afterwards he was successively manager of the Mysore West Gold Mining Company, and of the Consolidated Gold Mines of Western Australia. Returning to England in January, 1897, he died March 4th, after a brief illness, of pneumonia.

Mr. John H. R. Molson was a successful and wealthy business man of Montreal, and a liberal contributor to McGill University in that city, as well as other public institutions and enterprises.

William Rotthoff was a German by birth; came to this country about 1882; was engaged in the construction of the blast-furnaces at Braddock, Pa.; ultimately became assistant to Mr. Julian Kennedy, in which capacity he went to the Lucy furnaces, Pittsburgh, Pa. Subsequently he had charge of blast-furnaces in the South, and, still later, he returned to become superintendent of the Carrie furnaces, of the Carnegie Steel Company, a position which he retained until 1894, when he was engaged to construct and manage the Duquesne blast-furnaces of the same company. He left the Carnegie Company in 1896 to become superintendent of the furnaces of the Troy Steel and Iron Works at Troy, N. Y. Mr. Schwab, President of the Carnegie Company, writes that Mr. Rotthoff was a man of marked ability, both as engineer and as furnace-manager.

John Thomas, by whose death the Lehigh valley loses one of its most conspicuous captains of industry, was born at Yniscedwin, South Wales, September 10, 1829, and was a son of the late David and Elizabeth Thomas. When Mr. Thomas was a mere lad his parents decided to make their home in America, and on June 5, 1839, the family landed in New York. They at once located in Pennsylvania, and lived for a brief period in Allentown. Then they moved to Catasauqua, where Mr. Thomas spent his youth. He started in his career by entering the blacksmith shop of the Crane Iron Works, Catasauqua, as an apprentice. Later he worked in the machine-shops and at the furnaces. In the meantime his father had become superintendent of the Crane Iron Works; and upon his retirement he was succeeded by his son, John. Mr. Thomas filled the position with marked ability and success until 1867, when he resigned to accept the general superintendency of the Thomas Iron Company's works at Hokendauqua. Under Mr. Thomas's management the facilities of the works were almost doubled, and the product of the furnaces became known throughout the United States, and ever since the Thomas Iron Company has been a household word in the iron-trade. Advancing years and failing health compelled Mr. Thomas, in 1893, to relinquish the active management of the works, and he gave way to his son, David H. Thomas, who has been in charge ever since. Mr. Thomas was also largely identified with other business interests, as director of the Catasauqua and Fogels-

ville railroad, Catasauqua National Bank, Ironton Railroad Company, Morea Coal Company, Upper Lehigh Coal Company and the Pioneer Manufacturing Company of Thomas, Ala.

After the passage of a resolution instructing the Secretary to express by letter to the persons and corporations concerned the thanks of the Institute for courtesies received, the President declared the session and the meeting adjourned.

Members and Associates Elected.

Robert Allen,	Coolgardie, W. Australia.
J. W. Anderson,	Pittsburgh, Pa.
Rafael Maximo de Arozarena,	Pachuca, Mexico.
George J. Bancroft,	Denver, Colo.
Harvey Beckwith,	Philadelphia, Pa.
Frank Austin Bird,	Park City, Utah.
Richard H. Britt,	Silver City, Idaho.
E Percy Brown,	North Brookfield, Nova Scotia.
Henri Al. Cardozo,	Paris, France.
Frank Carpenter,	Pittsburgh, Pa.
George Clark,	Yalzoo, W. Australia.
Bernard T. Colley,	Argentine, Kan.
Herbert J. Daly,	Melbourne, Victoria, Australia.
W. H. Downey,	Yalzoo, W Australia.
Noah Fields Drake,	Tientsin, China.
John L Elliot,	Indé, Mexico.
Henry Rives Ellis,	San Francisco, Cal.
Clyde Milton Eye,	Arastra, Colo.
S. E. Fairchild, Jr.,	Philadelphia, Pa.
Francis Nicholas Flynn,	Argentine, Kan.
T. E. Fuller,	Coolgardie, W. Australia.
W. E. Garrigues,	Pittsburgh, Pa.
Edmundo Girault,	Pachuca, Mexico.
S. F. Goddard,	Nothingham, England.
Harry Denis Griffiths,	Auckland, New Zealand.
Jacob Cleveland Haas,	Greenwood, B.C., Can.
Richard Hamilton,	Kalgoorlie, W. Australia.
Louis M. Hardenburgh,	Houghton, Mich.
J. A. Leo Henderson,	Leipzig, Germany.
C. F Holbrook,	Philadelphia, Pa.
Frank S. Holloway, Jr.,	Marysville, Mont.
James Barnes Humphreys,	New York City.
Roberto Ipiña,	San Luis Potosi, Mexico.
Herbert John Jessop,	Chihuahua, Mexico.
Alfred E. Jessup,	Sparrow's Point, Md.
Robert B. Johnston,	Kingston-on-Thames, England.
George Edgar Ladd,	Rolla, Mo.
Frederick J. Leacey,	Victor, Colo.

John F. Lewis, ...	. Braddock, Pa.
William Henry McClintock,	. Sonora, Cal.
George Macfarlane, . .	. Charters Towers, Queensland, Aust'a.
Charles MacCulloch, . . .	. Melbourne, Victoria, Australia.
William B. Middleton, .	. High Bridge, N. J.
Alfred S. Miller, ...	. Moscow, Idaho.
Charles Mindeleff, . . .	. Pueblo, Colo.
Michael Mathias Minehan, .	. Argentine, Kan.
Milton J. Moore, ...	. Joliet, Ill.
James Morris, ...	. Boksburg, So. African Rep.
Francis James Oakes, Jr., .	. Johannesburg, So. African Rep.
José G. Palacios, ...	. Monterey, Mexico.
Arthur E. Pettit, ...	. Johannesburg, So. African Rep.
Herbert Pilkington, . . .	. Chesterfield, England.
Gifford Pinchot, ...	. New York City.
John Chester Ralston, . . .	. Spokane, Wash.
Ricardo Gomez Ramos, .	. Monterey, Mexico.
Armen Vaughn Raney, .	. Argentine, Kan.
J. W. Rayfield, ...	. Menzies, W. Australia.
Henry S. Reed, ...	. Deer Lodge, Mont.
Charles Rhodes, ...	. Auckland, New Zealand.
Pedro Pablo Rioseco, . . .	. Pachuca, Mexico.
J. A. Sanborn, ...	. New York City.
Dunbar D. Scott, ...	. Ironwood, Mich.
Karl Schmeisser, ...	. Clausthal, Germany.
Orville Campbell Skinner, .	. Pittsburgh, Pa.
E. S. Simpson, ...	. Perth, W. Australia.
K. N. Smith, ...	. Alden, Pa.
A. V. Stacpoole, ...	. Johannesburg, So. African Rep.
James Stanley, ...	. Coolgardie, W. Australia.
Burt C. Stannard, . . .	. Everett, Wash.
John Edward Stead, . . .	. Middlesborough, England.
James Underhill, ...	. Idaho Springs, Colo.
Charles G Van Brunt, . . .	. Kansas City, Mo.
A. Octavius Watkins, . . .	. Coolgardie, W. Australia.
W. H. Weed, ...	. Washington, D. C.
Arthur Raymond Weigall, .	. Batavia, Java.
John Whyte, ...	. Staten Island, N. Y.
Herbert Frank Widdicombe,	. Marysville, Mont.
Thomas J. Williams, . . .	. Moyston, Victoria, Australia.
Frederick Eaton Willson, .	. Keswick, Cal.
Nathaniel Wilson, . . .	. Johannesburg, So. African Rep.
Heneage Wynne-Finch, .	. London, England.
Hallett Winmill, ...	. Coolgardie, W. Australia.
Christian Sophus Wulff, .	. Johannesburg, So. African Rep.
Henry H. Yard, ...	. Philadelphia, Pa.
Charles Alexander Yorke, .	. Marysville, Mont.

ASSOCIATES.

V. B. Buck, Jr., ...	. New York City.
Nathaniel Curtis, ...	. Denver, Colo.
George McMurtrie Godley, .	. Boston, Mass.

Ella Knowles Haskell,	Helena, Mont.
C. W. Howard, Jr.,	Valley Springs, Cal.
W. E. Johnson,	Florence, Colo.
Joseph I. Kelly,	Sonora, Mexico.
Milo William Krejci,	Cleveland, Ohio.
C. R. F. H. Lamb,	Middlesex, England.
Curtis H. Lindley,	San Francisco, Cal.
Henry G. Maud,	Rangoon, Burmah.
William J. Parker, Jr.,	Cleveland, Ohio.
John R. H. Robertson,	Cana, Colombia, S. A.
Gustavus Sessinghaus,	St. Louis, Mo.
William Allen Smith, Jr.,	New York City.
Z. T. Sowers,	Washington, D. C.

CHANGE OF STATUS.

John T. Callaghan, Jr.,	Bethlehem, Pa.
A. D. Cross,	San Francisco, Cal.
B. I. Drake,	Bethlehem, Pa.
Joseph P. Gazzam,	Leadville, Colo.
C. F. Rand,	New York City.

The following persons were elected by postal ballot, November, 1897:

MEMBERS.

H. Foster Bain,	Des Moines, Iowa.
Thomas J. Barbour,	San Francisco, Cal.
Charles Lowthian Bell,	Middlesborough, England.
Arthur Hatfield Sumner Bird,	Salt Lake City, Utah.
Robert Macdonald Bird,	London, England.
John Blatchford,	Terry, So. Dak.
Hermann Brassert,	New York City.
William McC. Cameron,	Leadville, Colo.
Luis Campa,	Guanajuato, Mexico.
Hugo Carlsson,	Elyria, Ohio.
John S. Carnahan,	Sombrerete, Mexico.
Louis B. Carr,	Ouray, Colo.
Frederick H. Clymer,	Rockwood, Tenn.
Charles Frederick Courtney,	Broken Hill, New South Wales.
Sydney Cullingworth,	Cue, W. Australia.
William H. Davies,	Hazleton, Pa.
John Race Godfrey,	Sydney, New South Wales.
James Barrett Goodwillie,	Columbus, Ohio.
Charles B. Hall,	Pachuca, Mexico.
C. H. Hamilton,	Winnipeg, Manitoba.
Abbot A. Hanks,	San Francisco, Cal.
Erasmus Haworth,	Lawrence, Kan.
M. P. Gentry Hillman,	Birmingham, Ala.
James B. Laughlin,	Pittsburgh, Pa.
Frederick Ledoux,	Paris, France.
James I. Long,	Parral, Mexico.

Francis Arthur Mabris, . .	Cripple Creek, Colo.
Arthur Leggett Neale, . . .	Spitzkop P. O., Lydenburg District, So. African Rep.
Martin Nesbitt, ...	Chihuahua, Mexico.
Christopher Guy Orme, . .	Dublin, Ireland.
Harold Thomas Power, . .	Michigan Bluff, Cal.
Richard W. Rodda, . . .	Terry, So. Dak.
J. S. Schultze, ...	Boonton, N. J.
Harry R. Skinner, . . .	Roodepoort, So. African Rep.
Thomas Eddy Thomas, . .	Coolgardie, W. Australia.
Charles R. Longdale, . . .	Chihuahua, Mexico.
William W. Van Ness, . .	Buluwayo, So. Africa.
Arthur W. Warwick, . . .	Wickes, Mont.
Floyd Weed, ...	Cyanide, Colo.
Frank G. Willis, ...	Cripple Creek, Colo.
Robert A. Wood, ...	London, England.
Lewis Thompson Wright, . .	Keswick, Cal.

Associates.

Arthur Vincent Corry, . .	Golden, Colo.
F. D. Murray, ...	Newcastle-on-Tyne, England.
E. McGowan, ...	Newcastle-on-Tyne, England.
C. F. Rand, ...	New York City.

Change of Status.

John Goss, ...	Elyria, Colo.
John S. Pechin, ...	Cleveland, Ohio.

Excursions and Entertainments.

The privileges of the Casino were extended to members and guests of the Institute during the meeting, and, aside from the formal proceedings held in its beautiful hall, the pleasure of watching from its comfortable glass-enclosed corridor the Atlantic billows and surf was freely enjoyed.

On Thursday evening, February 17th, a subscription-banquet was held at the Hotel St. Charles. According to the uniform tradition of the Institute, these banquets are considered as private social reunions of ladies and gentlemen, secure alike from official record and from newspaper report. Those who attend them know their charm. For those who do not there is no way of getting the pleasure at second-hand. Yet the Secretary feels himself warranted in violating traditional privacy so far as to record that, on this occasion, the special feature was the presentation to Mr. Theodore D. Rand, of Philadelphia, of a testimonial in recognition of his twenty-five years' gratuitous

and cordial service as Treasurer of the Institute. This testimonial was given, as a personal tribute, by members of the Institute, represented by an informal committee, consisting of Messrs. E. W. Parker, John Birkinbine, Charles Kirchhoff and Geo. F. Kunz. The list of contributors was made up exclusively of those who had joined the Institute before 1880, or had been brought into personal contact with Mr. Rand through their attendance at meetings of the Institute, or through service upon its Council. It was presented with a graceful address, full of happy humor and reminiscence, as well as earnest cordiality, by President Drown, who had been, twenty-five years ago, the Secretary of the Institute, and upon whose personal request Mr. Rand had then consented to accept the responsibility of its treasurership. The testimonial was a single, superb crystal of beryl, pronounced by Mr. Rand himself (than whom there is no better authority) to be the finest he had ever seen.

On Friday morning, February 18th, a large party of members and guests left Atlantic City at 10 A.M., and proceeded to Philadelphia.

They were met at the ferry landing in that city by members of the Philadelphia Committee, and conveyed in carriages to The Philadelphia Museums, under the charge of Director W. P. Wilson. The peculiar character and function of these museums deserve a passing word of recognition here, although this is not the place for a detailed account of them.

The purpose of this institution is primarily commercial, namely, to familiarize American merchants and manufacturers with the products of other lands, and to indicate, as far as practicable, the demand and the opportunity for American products in those countries. In pursuance of this purpose, more than 100 rooms in the old Pennsylvania Railroad office building have been filled with really magnificent collections from nearly all parts of the world, showing what is produced in each country and the condition and form in which it is prepared and sent to market. The exhibits show also, in many cases, the goods imported into foreign countries, and the methods of packing and other peculiar requirements of trade, as now conducted. In connection with these most instructive exhibits a bureau has been established for the distribution of

detailed information, and the nucleus of a commercial library (which already assumes important proportions) has been formed. Arrangements are now in progress looking to an exposition in 1899 under the joint auspices of The Philadelphia Museums and the Franklin Institute. No doubt Director Wilson, whose office is at 223 South Fourth Street, Philadelphia, will take pleasure in answering inquiries from members of the Institute concerning this unique and most useful enterprise. Meanwhile, the present brief notice of it is amply deserved, both by the novelty and merit of the institution itself, and by the courtesy shown by its management to the Institute party.

In accordance with a cordial invitation from the Link-Belt Engineering Company, some of the party visited the shops of the company at Nicetown.

At 1.15 P.M. a lunch was given to the party at the Manufacturers' Club by the Philadelphia Local Committee.

MEMBERS, ASSOCIATES AND GUESTS REGISTERED.

The following list comprises the names of those who registered at hotel headquarters in Haddon Hall, together with those of the Philadelphia Local Committee, many of whom did not so register. There were in addition many members who attended the meeting, but, lodging at other hotels, did not record their names at hotel headquarters. And, in addition to all these, there were, accompanying members, about 60 ladies, whose names are not here given:

Taylor Allderdice.
Robert Allison.
W. S. Ayres.
H. P. Bellinger.
C. W. Bildt.
John Birkinbine.
R. C. Black.
C. P. Bower.
W. C. Brown.
William Burnham.
J. A. Capp.
H. A. Cardozo.
H. M. Chance.
Walton Clark.
J. Coan.
W. B. Cogswell.
F. C. Collingwood.
John H. Converse.
David T. Day.
W. B. Devereux.
James M. Dodge.
Benjamin I. Drake.
H. S. Drinker.
Thomas M. Drown.
Theodore Dwight.
Daniel Eagan.
Theodore N. Ely.
S. F. Emmons.
Thomas M. Eynon.
B. F. Fackenthal, Jr.
J. L. Ferguson.
B. E. Fernow.

F. Firmstone.
Stanley Flagg, Jr.
L. W. Francis.
Persifor Frazer.
John Fritz.
F. L. Garrison.
Stanley Gifford.
J. H. Harden.
G. W. Harris.
W. J. Harris.
John M. Hartman.
H. K. Hartzell.
C. Hewett.
G. C. Hewett.
J. T. Hilles.
Levi Holbrook.
H. M. Howe.
N. P. Hulst.
C. W. Hunt.
A. O. Ihlseng.
A. E. Jessup.
Guy Johnson.
J. E. Johnson, Jr.
T. D. Jones.
Washington Jones.
J. S. Kennedy.
W. Kent.
Charles Kirchhoff.
G. F. Knapp.
James C. Knox.
E. K. Landis.
A. R. Ledoux.
J. H. Lee.
W. L. Libbey.
J. Lilly.
A. S. McCreath.
William R. McIlvain.
C. A. Matcham.
W. K. Mitchell.
S. F. Morris.
W. H. Morris.
Charles E. Munroe.
W. G. Neilson.
E. E. Olcott.
G. Ormrod.
J. Pancoast.
E. W. Parker.
C. Q. Payne.
E. C. Pechin.
J. D. Pennock.
R. A. F. Penrose.
E. D. Peters, Jr.
Gifford Pinchot.
F. E. Platt.
T. D. Rand.
J. B. Randol.
R. W. Raymond.
William H. Rea.
R. H. Richards.
William H. Richmond.
Thomas Robins, Jr.
R. P. Rothwell.
R. W. Ryon.
Charles Schaffer.
H. Scuder.
H. J. Seaman.
H. G. Sears.
J. M. Sherrerd.
J. Bennett Smith.
Oberlin Smith.
W. R. Smith.
W. J. Taylor.
H. G. Torrey.
A. S. Van Wickle.
Leonard Waldo.
A. L. Walker.
John F. Wilcox.
W. H. Wiley.
John Wilkes.
W. G. Wilkins.
Oliver Williams.
Jones Wister.

Proceedings of the Seventy-Fifth Meeting, Buffalo, N. Y., October, 1898.

GENERAL COMMITTEE.*

W. C. Johnson, *Chairman;* E. B. Guthrie, H. J. March, T. Guilford Smith, Carl Meyer, E. C. Lufkin.

SUB-COMMITTEES.

TRANSPORTATION: Mr. Johnson.
LOCAL EXCURSIONS: Messrs. Smith, Meyer and Lufkin.
HOTEL ACCOMMODATIONS: Mr. March.
BUREAU OF INFORMATION: Mr. Lufkin.

BUFFALO ENTERTAINMENT COMMITTEE.

T. Guilford Smith, *Chairman;* P. H. Griffin, Maurice B. Patch, Eugene Roberts, Pemberton Smith.

HOTEL HEADQUARTERS: In Buffalo, the Iroquois Hotel; at Niagara Falls, the International Hotel.

The first session was held on Tuesday evening, October 18th, at the Buffalo Library Building, in the Lecture Room of the Buffalo Society of Natural Sciences. The meeting was called to order by Mr. T. Guilford Smith, Chairman of the Buffalo Reception Committee, and President of the Fine Arts Academy, who, after some appropriate words of greeting, introduced successively Hon. Conrad Diehl, Mayor of the city, Dr. Roswell Park, President of the Society of Natural Sciences, and Mr. Andrew Langdon, President of the Historical Society, each of whom cordially welcomed the Institute to Buffalo, and tendered to its visiting members and guests the attractions and facilities within their control. A suitable response to these addresses was made by President Kirchhoff, who then took the chair.

The Secretary made informal preliminary announcement of the death of three leading members of the Institute, reported since the last meeting, namely, Prof. Dr. Theodor Richter, of Freiberg, Saxony, Mr. Charles E. Emery, of New York City,

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* The arrangements for this meeting were kindly undertaken by the Engineers' Society of Western New York, of which Mr. W. C. Johnson, Niagara Falls, is President; and the General Committee consisted of members of that Society.

and Mr. Joseph C. Platt, of Waterford, N. Y.—of whom more detailed mention will be published in the *Transactions* hereafter.

The following paper, illustrated by lantern-views, was then read:

Note on Slips and Explosions in the Blast-Furnace, by F. B. Richards, Cleveland, Ohio.

After the close of the regular session, a number of remarkably fine lantern-slides, furnished by the courtesy of the Camera Club, of New York City, were exhibited by Mr. Theodore Dwight, to the astonishment and delight of the company, especially of those (apparently a majority) who were themselves more or less acquainted with the details and difficulties of photography, and to whom the results achieved in the taking of views under conditions generally deemed almost prohibitory were a suggestive revelation.

The remainder of the evening was spent in the inspection of the rooms and collections of the Society of Natural Sciences, the Buffalo Library, the Historical Society, the Fine Arts Academy, and the Society of Buffalo Artists, all of which are contained in the magnificent Library Building, and were hospitably open to the Institute, with the additional courtesy of attendants and guides, ready to point out and explain their manifold attractions.

The second session was held Wednesday morning, at the Iroquois Hotel.

The following paper was read and discussed:

The Kytchtym Medal, by Dr. Persifor Frazer, Philadelphia, Pa. (presented by the Secretary, in the absence of the author).

Interesting discussions of the subject of blast-furnace tuyeres,* and of the paper read at the preceding session by Mr. Richards, on "Slips and Explosions in the Blast-Furnace," occupied the remainder of the session, at the close of which the following papers were presented by the Secretary, in the absence of the authors, for discussion at the following session:

Modern Cupola Practice, with Special Reference to the Discussion of the Physics of Cast-Iron, by Bertrand S. Summers, Chicago, Ill.

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* The following papers formed part of this discussion:

Tuyeres in the Iron Blast-Furnace, by B. F. Fackenthal, Jr., Riegelsville, Pa.

Notes on Tuyeres in the Iron Blast-Furnace, by John M. Hartman, Philadelphia, Pa.

Note on the Forms Assumed by the Charge in the Blast-Furnace, as Affected by Various Methods of Filling, by Frank Firmstone, Easton, Pa.

The Secretary exhibited a remarkable specimen of crystallized gold, from the Mariposa Grant, California, presenting peculiar cubic and decahedric crystals of rudimentary skeleton-form.

The third session was held on Thursday morning at the same place.

The paper of Mr. Summers, presented at the preceding session, was discussed in a paper by F. E. Bachman, Buffalo, N.Y., on The Silicon-Control of Carbon in Cast-Iron, and in other written contributions and remarks.

The following papers were presented by the authors and discussed:

The Relations Between the Chemical Constitution and the Physical Character of Steel, by William R. Webster, Philadelphia, Pa.

A Description of the Semet-Solvay By-Product Coke-Oven Plant at Ensley, Alabama, by W. H. Blauvelt, Syracuse, N. Y.

The following paper was read by the Secretary, in the absence of the author:

The Effect of Sizing on the Removal of Sulphur from Coal by Washing, by Charles C. Upham, New York City.

The fourth session was held Thursday afternoon, at the same place.

The following paper was read by the author:

Corundum in Ontario, by A. Blue, Toronto, Canada.

The following papers were presented by the Secretary in the absence of the authors:

Graphic Records of the Screening of Crushed Materials, by Courtenay DeKalb, Kingston, Ontario, Canada.

Mill-Practice of the Utica Mills, Calaveras County, California, by W. J. Loring, Angels Camp, Cal.

A New Assay for Mercury, by Richard E. Chism, Mexico City, Mexico.

The Influence of Bismuth on Brass, and its Relation to Fire-Cracks, by Erwin S. Sperry, Bridgeport, Conn.

Experiments on the Sampling of Silver-Lead Bullion, by G. M. Roberts, Murchison Gold-Fields, Western Australia.

A Modification of Bischof's Method for Determining the Fusibility of Clays, as Applied to Non-Refractory Clays, and the Resistance of Fire-Clays to Fluxes, by H. O. Hofman, Boston, Mass.

Does the Size of Particles Have any Influence in Determining the Resistance of Fire-Clays to Heat and to Fluxes? by H. O. Hofman and Bradley Stoughton, Boston, Mass.

Hübnerite in Arizona, by William P. Blake, Tucson, Arizona.

The Superficial Alteration of Western Australian Ore-Deposits, by Herbert C. Hoover, Coolgardie, Western Australia.

Notes on the Mines of the Frontino and Bolivia Company, Colombia, S. A., by Spencer Cragoe, Mariana, Minas Geraes, Brazil.

Mineral Lode-Locations in British Columbia, by William Braden, Helena, Mont.

The following papers were read by title, for subsequent publication:

Note on the Cost of Tunneling at the Melones Mine, Calaveras County, California, by W. C. Ralston, San Francisco, Cal.

Note on the Operation of a Light Mineral Railroad, by James Douglas, New York City.

The Alluvial Deposits of Western Australia, by T. A. Rickard, Denver, Colo.

The Evolution of Mine-Surveying Instruments, by Dunbar D. Scott, Ironwood, Mich.

(This important paper was announced as a special subject for discussion at the next meeting).*

Analysis of Blast-Furnace Gas while Blowing-In, by Ralph H. Sweetser, Everett, Pa.

The International Correspondence Schools, Scranton, Pa., by H. H. Stock, Scranton, Pa.

Communications received, in discussion of the following papers of previous meetings, were also announced by the Secretary for future publication:

* This discussion (mostly in communications to the Secretary or the author) has proved so voluminous, and has covered so long a period, that its final publication cannot be made in the present volume. It will be included in vol. xxix., but sent to members in pamphlet form long before that volume is issued.

Mr. Keller's paper on the Elimination of Impurities from Copper-Mattes (Atlantic City Meeting, February, 1898).

Dr. Don's paper on the Genesis of Certain Auriferous Lodes (Chicago Meeting, February, 1897).

Prof. Howe's paper on the Use of the Tri-Axial Diagram and Triangular Pyramid for Graphic Illustration (Atlantic City Meeting, February, 1898).

Mr. Heath's paper on the Electrolytic Assay as Applied to Refined Copper (Lake Superior Meeting, July, 1897).

Dr. W. B. Phillips, Pittsburgh, Pa., presented a Note on the Possible Origin of the Pneumatic Process of Making Steel.

The Secretary announced that the Council had decided to hold the next meeting (beginning February 21, 1899) in the city of New York.

The fifth and final session was held on Friday evening, October 21st, at the International Hotel, Niagara Falls.

Mr. W. C. Johnson, Chairman of the Local Committee, made an informal address, illustrated with the lantern, explaining the existing works for the utilization of water-power at Niagara Falls.

The meeting was then adjourned; and after adjournment, Mr. Dwight exhibited, by request, for the benefit of members who had not been present at the first session, some of the lantern-slides which had attracted so much interest and admiration on that occasion.

Members and Associates Elected.

The following persons were elected members or associates at this meeting:*

Members.

Robert Addie, ...	Ellesmere Port, Cheshire, Eng.
Joseph S. Atkins, . . .	Glenville, Ohio.
Alfonso Z. Baldenebro, . .	Mexico City, Mexico.
Jean Le Bret, ...	Paris, France.
J. A. Brinell, ...	Fagersta, Westanfors, Sweden.
James C. Davis, . . .	Chicago, Ill.
Newton A. Dunyon, . . .	Eureka, Utah.
Stanly A. Easton, . . .	Silver City, Idaho.

* A very large number of proposals received by the Council were reserved for future action; it being impossible for the Council to consider them with proper deliberation during the meeting.

Thomas M. Eynon, . . .	Philadelphia, Pa.
Frank J. Fitzsimmons, . .	Harqua Hala, Arizona.
Ernest Grave, ...	Chihuahua, Mexico.
Percy Grave, ...	Chihuahua, Mexico.
Herbert E. T. Haultain, . .	Rossland, British Columbia.
P. Y. Heckman, . . .	Issaquah, Washington.
Benjamin Hodge, . . .	Fort Steele, British Columbia.
John W. Hosking, . . .	Wilkeson, Washington.
Harry R. Kimmel, . . .	Cleveland, Ohio.
Frank Langford, . . .	Tobacco, Montana.
H. B. Lawrence, . . .	Ocampo, Chihuahua, Mexico.
Thomas W. Lawson, . . .	Baltimore, Md.
Irving S. Lydecker, . . .	New York City.
Dorsey A. Lyon, . . .	Stanford University, Cal.
James A. McClurg, . . .	Denver, Colorado.
Leslie C. Mott, ...	Chihuahua, Mexico.
Carl A. Netto, ...	Frankfort on Main, Germany.
William A. Prichard, . .	Stanford University, Cal.
John H. Swoyer, . . .	Sierra Mojada, Coahuila, Mexico.
Samuel C. Thomson, . . .	Johannesburg, South African Rep.
Alfred Tilly, ...	Cornwall, England.
Howard F. Wierum, . . .	Pueblo, Colorado.

Associates.

Tatsuzo Kosugi, ...	Azabu, Tokyo, Japan.
Rodrick McKenzie, . . .	Westport, New Zealand.
William Morris, . . .	Greymouth, New Zealand.
E. Zintgraff, ...	Hoboken les Anvers, Belgium.

Excursions and Entertainments.

Wednesday afternoon, October 19th, was devoted to a general excursion by steamer through Buffalo Harbor, presenting a view of the important improvements in progress, under the direction of both the local authorities and the United States Government, for the purpose of perfecting the commercial facilities of this important lake port. A visit to the Great Northern Elevator, one of the largest in the United States, which is entirely operated by electric power, transmitted from Niagara Falls, and an inspection of the Union Iron Works, and the blast-furnaces of the Buffalo Furnace Co., were included in this excursion.

On Thursday evening, October 20th, a delightful social reception, with music, dancing and supper, was given by the Buffalo Entertainment Committee to members and guests of the Institute at the rooms of the Ellicott Club.

Friday was wisely left free for optional excursions of indi-

viduals or small parties to the various local works and points of interest, arrangements for suitable reception having been made and guides provided by the Local Committee.

Among the establishments hospitably thrown open to such visits were the following:

The Buffalo Railway Company (now using power from Niagara Falls).

The Buffalo Smelting Works (smelting copper from Lake Superior).

The Buffalo Furnace Company (manufacturers of foundry and Bessemer pig-iron).

The Union Iron Works (under construction).

The Buffalo Pitts Company (manufacturers of steam road-rollers, traction-engines and threshing-machines).

The New York Car-Wheel Works (manufacturers of cast-iron car-wheels).

The P. H. Griffin Machine-Works.

The Buffalo Bridge and Iron Works.

The Pratt and Letchworth Company (manufacturers of malleable iron and steel castings).

The Snow Steam-Pump Works (manufacturers of steam-pumps of all sizes).

The Buffalo Structural Steel Company.

The Wagner Palace-Car Company.

The People's Gas-Light and Coke Company.

The Buffalo Forge Company (manufacturers of electric-light engines, fans, blowers, forges, etc.).

The pumping-station of the Bureau of Water of the Buffalo Department of Public Works (the largest pumping-station in the world under one roof, containing seven engines with a daily capacity of 20 million gallons each, and two vertical triple-expansion engines with a daily capacity of 30 millions each).

The Lake Erie Engineering Works (builders of the two 30-million gallon engines mentioned above, and manufacturers of steam-engines of all sizes, gun-carriages, boilers, etc.).

Also the following works, located at Depew, near Buffalo:

The Union Car Company (manufacturers of freight-cars and car-wheels).

The Gould Coupler Company (manufacturers of couplings, axles, etc.).

The New York Central and Hudson River Railroad Locomotive-Shops.

Saturday was spent at Niagara Falls in the enjoyment of the scenic beauties of the place and in visiting the power-houses of the two companies now utilizing the Niagara water-power, and the interesting works of the Carborundum Company.

A large party remained at the Falls over Sunday, the International Hotel, which would otherwise have closed on Saturday, being kept open for their accommodation. The quiet enjoyment of nature and the social intercourse among friends thus made possible will remain in the memories of many as one of the chief delights of this very successful meeting.

Members, Associates and Guests Registered During the Meeting.

The following list, though not absolutely complete, comprises the names of those who took the pains to register themselves at hotel headquarters in Buffalo. A number joined the party at Niagara Falls. A large proportion of the members registered were accompanied by ladies, whose presence, it need not be said, though not matter of official record, added greatly to the social success of the meeting:

Taylor Allderdice,
Philip Argall,
F. E Bachman,
Frank Baird,
F. V. E. Bardol,
R. T. Bayliss,
H. P. Bellinger,
W. H. Blauvelt,
A. Blue,
F. E. Bodwell,
H. T. Buttolph,
J. A. Capp,
Edgar S. Cook,
Torbert Coryell,
W. C. Davis,
G. L. Davison,
Courtenay DeKalb,
W. F. Downs,
Thomas M. Drown,
Theodore Dwight,
Thomas M. Eynon,
W. E. C. Eustis,
B. F. Fackenthal,
Austin Farrell,
J. W. Fuller, Jr.,
Oliver S. Garretson,
James Gayley,
Fritz Gleim,
P. H. Griffin,
E. B. Guthrie,
John M. Hartman,
Gus C. Henning,
C. T. Holbrook,
W. G. Houck,
George S. Hubbell,
George S. Humphrey,
W. S. Hungerford,
W. C. Johnson,
Edward Keller,
Charles Kirchhoff,
G. F. Knapp,
James F. Lewis,
John Lilly,
Ralph Lowe,
George F. Lucas,
E. C. Lufkin,
Frank Lyman,
James A. McClurg,
William R. McIlvain,
H. J. March,
Charles A. Matcham,
Carl Meyer,

C. M. Morse,
Charles T. Neal,
H. L. Noyes,
George Ormrod,
I. P. Pardee,
E. W. Parker,
William B. Phillips,
A. E. Piorkowski,
A. C. Rand,
J. B. Randol,
R. W. Raymond,
F. B. Richards,
R. T. Schraubstadter,
H. J. Seaman,
J. M. Sherrerd,
Bennett Smith,
I. Bennett Smith,
Pemberton Smith,
T. Guilford Smith,
E. C. Sorhborger,
George Somers,
H. H. Stoek,
Ralph H. Sweetser,
F. M. Sylvester,
Howard V. Thomas,
Alfred W. Thorn,
B. W. Tichenor,
J. C. Ulmer,
T. B. Walker,
William R. Webster,
Charles H. Wellman,
John F. Wilcox,
Charles Wiley.
William H. Wiley,
E. H. Williams,
Oliver Williams.

PAPERS.

Geological Excursion Through Southern Russia.

BY S. F. EMMONS, WASHINGTON, D. C.

(Atlantic City Meeting, February, 1898.)

THE seventh session of the International Congress of Geologists, held in Russia during the past summer, was an epoch-making event. The last Emperor had given his personal sanction to the invitation extended by the Russian geologists to the Congress during its meeting at Washington in August, 1891, and the present Emperor had nobly carried out the intent of that sanction, which, as our Russian colleagues then assured us, meant that the Empire would be thrown open to our inspection, and that the expenses of our journeys through it would be borne by the government. Nor was this all; for, by the unwearied exertions of the General Secretary, Mr. Theodore Tschernycheff, who had given up his time for the past two years to this work, the whole Russian people had been so aroused to the importance of this occasion, when geologists would come from all parts of the world to view the natural wealth of their country, that we were everywhere received with the most lavish hospitality.

Fine club-houses in the larger cities were not only given up to our use during the day time, but were often fitted out with beds for the night, so that we could be independent of the often rather indifferent hotel-accommodations. We were banqueted on every possible occasion, and often several times on the same day. Wherever it was, in the summer palace of the Czar at Peterhof, in city halls and nobles' clubs of the larger cities, in the improvised banqueting-halls at the mines, made for the occasion and dressed with flags and triumphal arches, or in the hospices on the summit of the Caucasus, which furnish to the ordinary traveller only a shelter for the night, we found the tables groaning with the choicest viands and wines that the Empire afforded, and served by a host of waiters which had apparently followed us from the capital. Nor did these warm-

hearted people content themselves with merely ministering to our physical well-being. Addresses of welcome and complimentary speeches abounded on every occasion—after-dinner speeches they could hardly be called, for they commenced with the fish and continued often into the small hours of the night, accompanied by abundant libations. This had its disadvantages; for, as a matter of courtesy, some one of us had to respond in equally complimentary vein, and after a week or two of this kind of thing it got to be a pretty severe task for those to whom this duty fell to find something new to say on each successive occasion, especially when it had to be said in language foreign to the speaker. Fortunately the linguistic accomplishments of our successive hosts seemed to decrease as the distance from their capital increased, and speeches of welcome in the Russian tongue soon became the order of the day. We were not slow to take the hint, and retorted in our own language—English, Swedish, Norwegian, Bulgarian, French, German, or even Japanese, as the case might be—happy in the consciousness that our hosts could not tell that we were dealing out to them the same old "chestnuts" that had served on many previous occasions.

Russia has evidently been waking up to a realizing sense of the importance of developing its great mineral wealth, even if this has to be done by foreign capital. Leaving out of consideration the relatively small number of its highly-educated citizens, the mass of the people has been hitherto very exclusive, and has looked with distrust on foreign innovations. Within the past few decades, however, much Belgian, English, French, German and Swedish capital has been invested in Russian mines and oil-fields, and the resulting industrial prosperity has been felt by all classes. There was probably, therefore, a legitimate element of self-interest in the warmth of our reception; the masses evidently believed that our visit would further increase industrial development, while the scientific men openly avowed their expectation that it would result in more liberal endowments for scientific research on the part of their government.

Of the scientific sessions of the Congress, held in St. Petersburg during the week commencing August 29th, I do not propose to speak, nor of the excursions through Finland and the Urals, which took place previous to these sessions, and which I

was unfortunately unable to attend. Immediately after the close of the sessions, those of us who proposed to take part in the southern excursion, to the number of about 300, proceeded to Moscow in special trains provided for our use. Each visiting member of the Congress, I may remark, had received before reaching the country a ticket which entitled him to first-class passage over all the railroads in the Empire during the months of July, August, September and October. At Moscow we were divided into three parties, each following a different route to the Caucasus. One went eastward to Nijni Novgorod and descended the Volga by steamer; a second proceeded southwest to the ancient city of Kiev, on the Dnieper, while the third took an intermediate southerly course, through the University city of Kharkow and the basin of the Donetz, to Rostow, at the mouth of the Don, on the Sea of Azof. All three parties finally centered at Vladikavkaz, on the north slope of the Caucasus, at the mouth of the gorge of the Terek, the northern end of the famous military road that crosses the mountains to Tiflis in Transcaucasia, thus connecting Europe with Asia. From Tiflis a long special train took the reunited geologists, first eastward to Bakou on the shores of the Caspian, and then back again westward to the Black Sea at Batoum, the port latest transferred from Turkey to Russia. Here our party, now reduced to something under 200 by the departures of those who had gone on side-excursions to Mt. Ararat and other places, embarked on a new English-built steamer, the Grand Duchess Xenia. On this steamer, which had been engaged for this especial service, and provided with a chef and an excellent corps of waiters from Odessa, we spent ten most delightful days, cruising along the eastern and northern shores, landing frequently for geological excursions along the precipitous and picturesque southern coast of the Crimea (which has been compared, with good reason, to the Riviera of Italy), and occasionally going out into the depths to make soundings and dredgings. After a few days spent at Sevastopol, visiting the battle-fields and many points of interest in the neighborhood, we crossed in a fine summer storm to Odessa, the great commercial port of the Empire, and, after a final banquet on October 6th, given by the municipality, telegraphing our heartfelt thanks to the Czar and the Grand Duke Constantine, the special patron of

science in Russia, we broke up and departed to our respective homes.

It would evidently exceed the limits at my disposal to describe all that we saw during the month thus spent, and I shall only attempt to present a hasty sketch of a few of the features of our trip that would particularly interest the mining engineer and geologist. Such a sketch, moreover, must necessarily be very superficial; for only a Russian is equal to making a practice of feasting till midnight and commencing his geological work with a clear head at six o'clock in the morning; and, excellent though our Russian guides were, it was evidently beyond even their capacity to answer all the questions put to them by the 80 or more inquiring geologists that each of them had charge of.

The Basin of the Donetz.

The route followed by the party to which I was attached, from Moscow to the northern flanks of the Caucasus, a distance in a straight line of about 850 miles, gives a good general idea of the geological structure of the great plains of Eastern Europe, the original home of the Russian race. In its geological structure, as in many other features, Russia is said to resemble the United States, and certainly does so, more than does the rest of Europe; but in this, as in other respects, to a visiting American, the contrast between the two countries is very strong. With us the different geological provinces are, for the most part, clearly marked and distinct, and it is a comparatively simple task to decipher the broad general outlines of their structure. As Sir Archibald Geikie has recently remarked:*

> "Had the study of the earth begun in the New World instead of the Old, geology would unquestionably have made a more rapid advance than it has done. The future progress of the science may be expected to be largely directed and quickened by discoveries made in America, and by deductions from the clear evidence presented on that continent."

In Russia (except the Urals and the Caucasus on its eastern and southern borders) there are neither well-defined mountain-uplifts nor deep cañons along the rivers that lay bare the relations of the older rocks. In the intermediate plain-country the

* *The Founders of Geology.* Macmillan & Co., N. Y., 1897, p. vii.

older stratified rocks, and even the Tertiary beds, have been abundantly plicated; but the folds have been planed down without leaving any decided impress upon the present topography, and their outcrops have been largely covered by a mantle of Quaternary deposits. Hence the geologist is rarely able there, as with us, to trace the successive formations by actual inspection and lithological correspondence, but must depend mainly on occasional finds of fossils and on deep drill-holes.

The development of the Donetz basin, whose abundant mines of coal, mercury and other minerals now make it one of the most important industrial regions of the Empire, is a matter of quite recent date. Carboniferous rocks have indeed been known to exist there for a long time, but it is only since 1892 that the Russian geological survey has undertaken to furnish a basis for the systematic development of its mineral wealth by making detailed geological maps of the region, which are yet far from completion.

This so-called basin is a sort of rolling plateau, treeless, but well-grassed, having a maximum elevation of 1200 feet above sea-level. The actual exposures of Carboniferous rocks cover an area of less than 8000 square miles, but the extent of available coal-seams will undoubtedly prove much greater when the geological structure is better known. The coal-seams, which are very numerous, but, in general, rather thin, occur in the lower part, though not at the base, of the Carboniferous system. The character of the sediments and their fauna present remarkable analogies with the Coal-Measures of the Mississippi basin. A geological section of over 3000 feet of strata gives less than thirty workable seams of coal, most of which are under 3 feet, and none over 6 feet, in thickness. Single mines exploit up to eight or nine seams, with an aggregate thickness of 14 to 16 feet of coal. The coals themselves present all varieties, from anthracite through coking- and gas-coals to a light bituminous coal which is not sufficiently compact to bear transportation.

The Carboniferous rocks contain also deposits of gold, mercury, silver, lead, zinc and iron; but although the existence of gold, lead and zinc has been known for over 100 years, and attempts have been made from time to time to mine them, they have not been proved as yet to possess much economic value.

They occur in thin quartz-veins, in the breccia of faults, and in fracture-zones traversing the strata, generally in a vertical direction, and sometimes in cavities between the strata near such faults. The country-rocks are Carboniferous sandstones, limestones and slates. Specular iron-ores occur at the outcrops of certain limestone strata, extending down only about 30 feet below the surface, but do not give promise of any great industrial importance. The most important metallic product is that of the mercury-mine near Nikitovka, whose output is about a million pounds per annum. It was discovered in 1879 by A. Minenkow, a Russian mining engineer. The deposits occur in quartzites associated with micaceous sandstones, shales and a few thin coal-seams, all of Carboniferous age. The principal ore is cinnabar, with stibnite and pyrite as associated minerals. These occur impregnating a friction-breccia of quartzite inclosed in veins which have sometimes two well-defined and polished walls, sometimes only a single wall. In the latter case, there are wide zones of ground-up quartzite, impregnated with cinnabar, which are mined in open cuts. These veins generally occur near the crest of anticlinal folds, or "coupoles" as they are called. The principal lines of fissures are parallel to the axes of these folds, but there are systems of cross-fissures later than these. The Russian geologists consider that the carbonaceous contents of the rocks have influenced the concentration of the mercury, and accept, as applicable here, G. F. Becker's explanations of the manner of deposition of mercury from solution. In some cases the neighboring coal-seams inclose distinct and perfectly-formed crystals of cinnabar.

Rock-Salt.—In the northwestern corner of the Donetz basin, near Bakmout, are deposits of gypsum, anhydrite and rock-salt. Salt-springs had long been known there; but it was not until a drill-hole, sunk in 1874 under the direction of M. Karpinsky, of the Geological Survey, had disclosed, in a depth of 764 feet, an aggregate thickness of 343 feet of rock-salt, that mining was commenced. The rock-formations exposed consist, in the upper part, of clays, marls, gypsum and anhydrite, with salt at the base, belonging to the Middle Permian, and, in the lower part, of alternating beds of dolomite and salt of the Lower Permian. The principal mine, that of Briantevska, which we visited, belongs to a French company. The bed worked is 117 feet thick

and 300 feet below the surface, being near the base of the Middle Permian. The mines are lighted by electricity, and the apparently endless succession of great chambers, 50 to 100 feet high, and many hundred feet long, hollowed out of white glistening rock-salt, form a most impressive sight, not surpassed by the famous German salt-mines at Stassfurt. They have found as yet, however, none of the potash-salts which constitute so important a part of the wealth of the latter deposits. The product of the salt-mines in the neighborhood, which have been opened scarcely 16 years, is said to amount to over 288,000 tons annually.

Evidently the basin of the Donetz is destined to become one of the great industrial centers of Russia, and to rival in the value of its product the famous metalliferous region of the Urals. In our hasty visits to a few of the many great coal-mines we could form no idea of the aggregate product. In 1883 the coal-product of the Donetz basin was three-fourths of that of the entire Empire, and in 1893 the product of the Empire had increased from a quarter of a million to eight million tons, most of which was probably from the Donetz. The same rock-formations apparently underlie all central Russia between St. Petersburg and the Urals, and other coal-basins may be discovered and opened in the future; but the association of useful minerals in this limited region is most remarkable, and presents great industrial possibilities.

The Caucasus Mountains.

The most imposing scenes of our trip, from a geographical and ethnological point of view, were in the great mountain chain of the Caucasus, 800 miles long, which separates Europe from Asia. It has at least four peaks over 16,000 feet high (or higher than Mt. Blanc, which we are accustomed to consider the highest mountain in Europe), the most prominent of which, Mts. Elbruz (18,525 feet) and Kasbek (16,546 feet), have been built up by comparatively recent eruptions of igneous rock. The main part of the chain is less than 100 miles wide, but the general uplift extends into the Armenian plateau to the south, in branching ranges, called by the Russians the Anti-Caucasus. The interior of the main chain does not appear to have developed as yet any important mineral deposits; and the part that

we saw during our few days' journey across it by the famous "Military road to Georgia" showed very little of what the western prospector calls "mineral sign." It would be premature to say, however, that it is not likely to prove rich in mineral resources in the future, for it has as yet been but only superficially explored from a geological standpoint, and the average Russian considers it a dangerous region to travel in. To the ethnologist, however, it is teeming with interest, being inhabited by nearly a dozen different races, more or less radically diverse in language, idiom or religion, of whose origin little is yet definitely known.

On either flank of the chain, however, the Russians have made considerable developments since their conquest of the fierce mountain tribes that used to prey upon the surrounding country.

Toward the western end of the northern flank are the Thermal Springs of the Caucasus, now the most famous in all Russia. Around these within the past twenty years have been built up a number of handsome towns, supported by those who come from all over Russia to take the waters. The principal of these are Piatigorsk, Jeliéznowodsk, Essentouki and Kislowodsk. The springs are situated in the midst of a picturesque group of conical hills, rising out of the Cretaceous and Tertiary plains, which are formed by laccolitic intrusions of trachyte. The spring at Kislowodsk, which is nearest the base of the mountains, is an effervescent spring called Narzan, said to have been known in the time of Peter the Great, and visited by the geologist Pallas in 1792. It is the Apollinaris water of Russia, and not at all unlike this water in taste. From Kislowodsk, a night-drive up the mountains of 40 versts brings one at sunrise to Bermamout (8550 feet), a flat ridge of Jurassic limestone from which one obtains a glorious view of the snow-covered mass of Mt. Elbruz.

Still further east, beyond Vladikavkaz, are the oil-fields of Grozny, of quite recent development. They occur, like those on the south flank, in Oligocene Tertiary beds; the oil and gas come to the surface most abundantly along the sides of anticlinal folds. They are said to give promise of a large production, a single flowing well having sent out in a year and a half 1,440,000,000 pounds of oil.

From the northern to the southern slopes of the main chain of the Caucasus there is a marked change in both physical and geological conditions. The northern slopes of the mountains are practically treeless, and slope somewhat gently toward the broad grassy plains of the Government of Stavropol, which in many respects resemble our Great Plains, and, like them, are better adapted for pastoral than for agricultural pursuits. The underlying rock-formations, moreover, appear to be but little disturbed, and occupy in general a nearly horizontal position. The southern slopes are steeper and more broken, and, below timber-line, support an abundant forest-growth; the climate is of course warmer, the valleys are fertile and produce varied and abundant fruits, especially of the vine, which makes a wine very similar to that of California.

The underlying rock-formations, mostly Tertiary, are sharply folded and faulted, forming an extremely broken country of irregular ridges and smaller mountain chains, which extend beyond the valley of the Koura into the volcanic plateau of Armenia. The culminating point of this plateau is the beautiful isolated volcanic cone of Mt. Ararat, over 17,000 feet high, which rises from a base scarcely 3000 feet in altitude, on the present southern boundary of the Russian Empire. To this some members of our party made a pilgrimage, the pleasure of which was clouded by a fatal accident to one of the scientific guides, in his attempt to reach the summit before his companions.

In this Transcaucasian country, manganese- and copper-mines of considerable value have already been opened, and the geological conditions seem favorable to the development of other ores; but its great wealth at the present day lies in the enormous supplies of petroleum, of which I shall next speak.

Through it, from northwest to southeast, runs the valley of the Koura, which has its sources in the mountains to the north and south, and, at a point little over 50 miles from the Black Sea bends abruptly and flows eastward for 450 miles into the enclosed basin of the Caspian Sea, whose level is 3 meters lower than that of the Black Sea. It thus forms, between these two basins, a natural highway, which has been taken advantage of by the Russian Government to build its railway, opening the oil-fields to the rest of Europe.

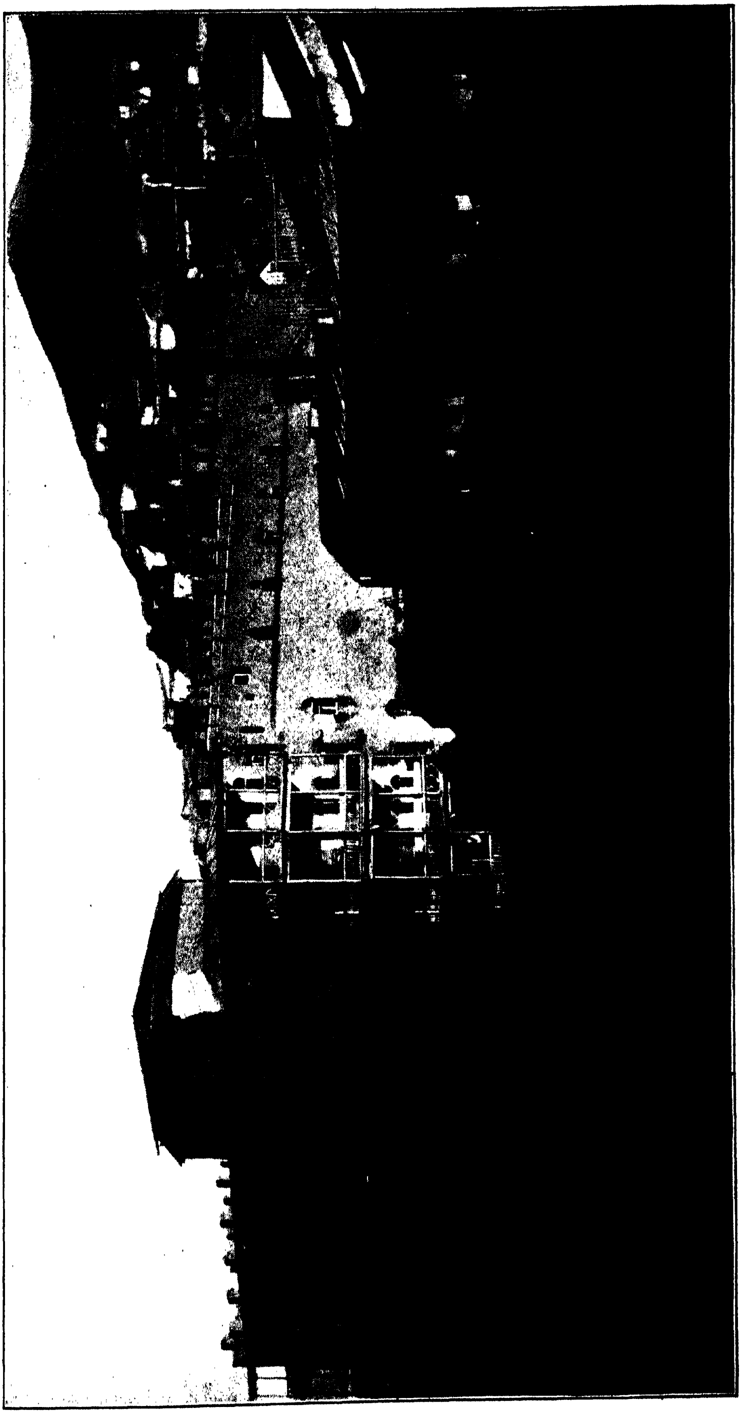
Midway between the two basins, and occupying both banks of the Koura, lies the picturesque and highly interesting city of Tiflis, with 159,000 inhabitants, said to speak 70 different languages. It has had a very checkered history. It became the capital of the Georgian Empire at the beginning of the 6th century, and was pillaged by Tamerlane in 1395. Since then it has been at times under Persian rule, but finally passed into Russian control with the consent of its original owners, the Georgians. The greater part of its present population is either Georgian, Armenian, Russian or Persian. Fig. 1 represents a bit of the old town seen from the Avlabarsky bridge over the Koura.

Petroleum Deposits of Baku.

About sixty miles from the mouth of the Koura the railroad leaves that valley and crosses in a northeast direction a dreary bit of a clayey desert, utterly barren of any vestige of vegetation, toward the peninsula of Apcheron, on the southern flanks of which outcrop the oil-measures of Baku. This peninsula, which projects nearly 50 miles into the Caspian, is on a direct line with the main axis of the principal chain of the Caucasus. The older rocks which form the core of that chain are no longer visible here, but are buried beneath the Oligocene Tertiary beds, which outcrop for long stretches on either flank. At many other points these beds give evidence of being oil-bearing, one of the common signs being the little mud-volcanoes, so-called, through which there is an escape of natural gas. I do not propose to give any extended or technical account of these world-famous petroleum deposits, of the fires dancing on the waves of the Caspian in the bay of Baku, when one touches a match to the natural gas escaping through them, or of the old Parsee temple in the desert, whither the ancient fire-worshippers made yearly pilgrimages to see the eternal fires fed from the same source. These things have already been fully described by those who are more familiar with the subject.

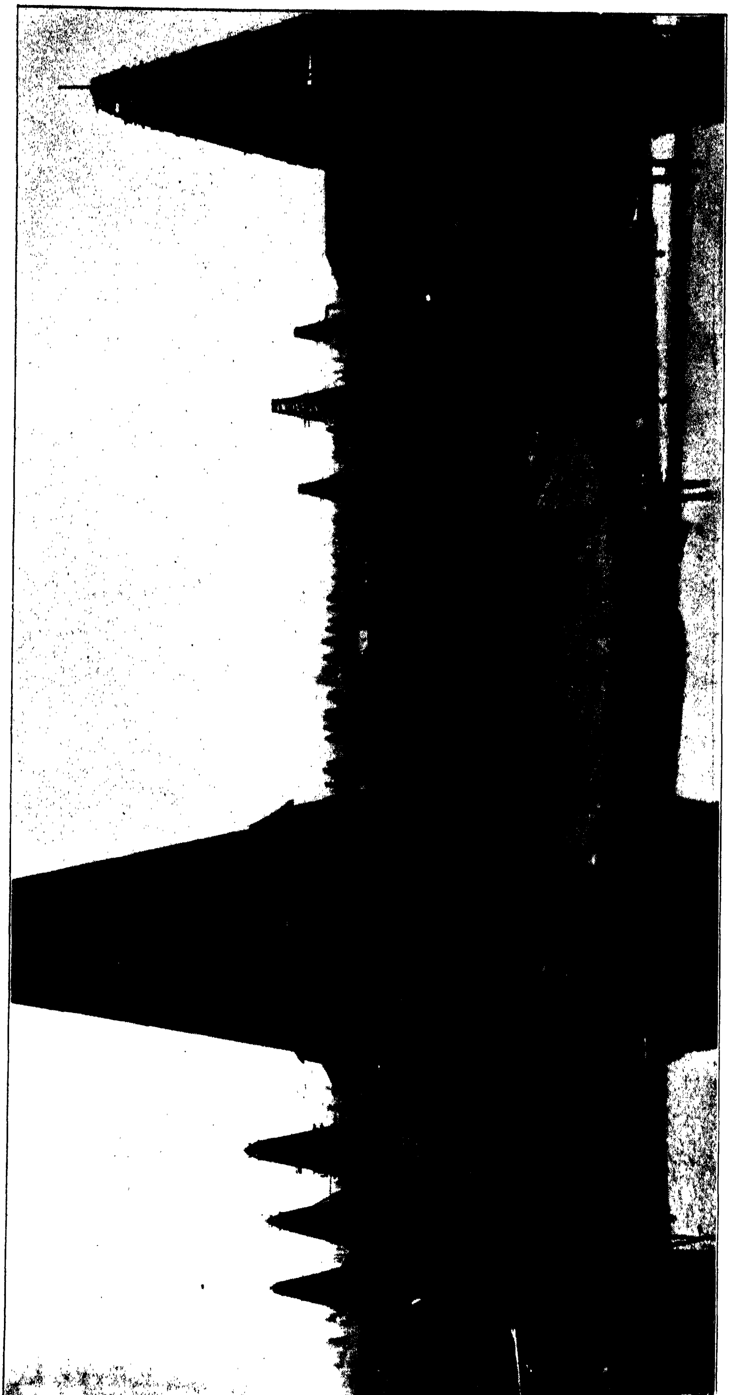
Baku, in spite of its dreary surroundings, is a most interesting place, and the views from it out upon the blue waters of the Caspian are most picturesque. It is built upon the site of an old Tartar town dating from the 16th century; but the present city, which has more than 120,000 inhabitants, has

Fig. 1.



Tiflis from the Avlabarsky Bridge over the Koura.

Fig. 2.



Oil-wells at Balakhany near Baku.

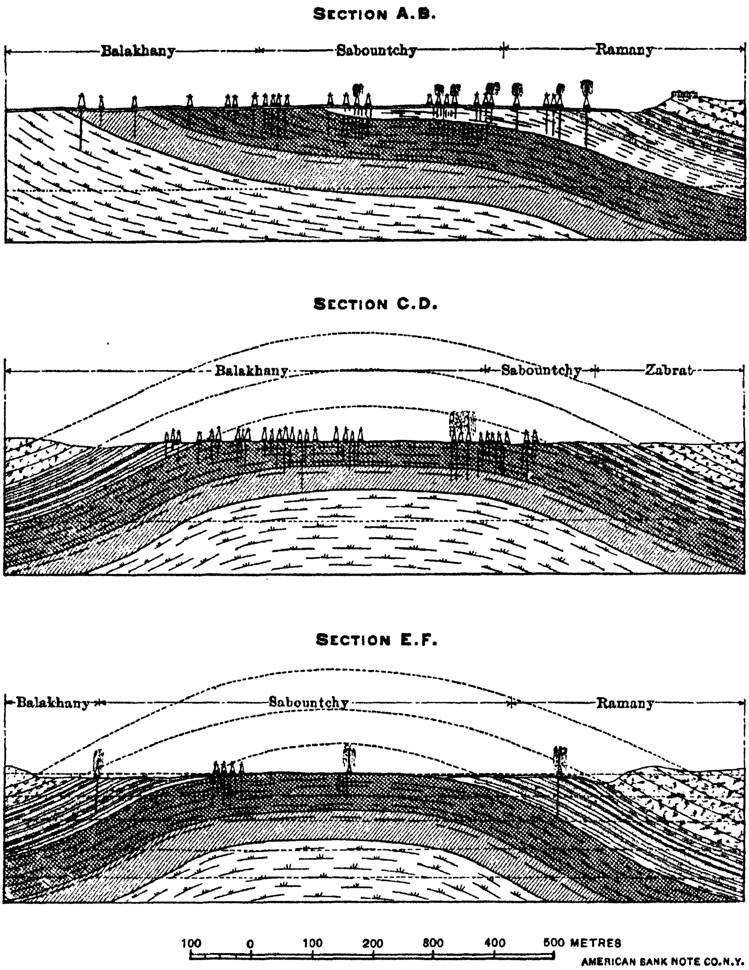
sprung up within the comparatively few years since the Swede, Nobel, commenced the production of petroleum on a large scale. In spite of the many strong contrasts which first strike the eye, in the people, the buildings and the surroundings generally, Baku has many points of resemblance with the oil-fields of Pennsylvania. The richest citizen is a big rough-looking Tartar, who, a few years since, was a common mason, working for day-wages. Oil-derricks have a generic resemblance all over the world. The view of some of those at Balakhany, the principal oil-field, 6 miles out of Baku, given in Fig. 2, might answer also for those I saw recently on the west coast of Peru.

The central derrick in Fig. 2 is what is called at Baku a fountain, but would in our oil-well parlance be a "gusher." The pool in the foreground is oil, which overflows from the well every few minutes. The action of such a well is most interesting to watch, and resembles closely that of the intermittent geysers of the Yellowstone Park, except that instead of pure white water, it sends up a column of a greasy, greenish-brown fluid of most repulsive appearance. Between the discharges the oil is pumped up and into a pipe-line, leading to the wharves at Baku, six miles distant. At intervals of about five minutes, oil commences to flow over the lip of the well, then rises with a dull roar in a perpendicular jet, at first only a few feet, but with a recurring motion, as if forced up by the stroke of the piston of a pump, at each stroke rising a few feet higher, until it finally reaches the top of the clap-boarded shack which surrounds the derrick, and spouts through the top, pouring a greasy flood down the slanting sides, and running off in a small brook into the rude cistern, a simple depression in the clayey soil, arranged to receive it. Then it gradually subsides as it rose, rising at each stroke of the imaginary piston not quite as high as at the preceding; finally all becomes quiet again, the men in charge of the pumps, dripping from their shower-bath of oil, resume their stations, and the pumps go on as before.

The geological structure of the beds and the conditions ot flow seem also to fit the anticlinal theory so ably expounded for our American oil-fields by Prof. Orton. There are the same outpourings of natural gas, oil and salt water, and apparently in the same relations of specific gravity, as in Ohio. The anti-

clinal structure is well shown by the sections given in Fig. 3, which are copied from the admirable guide-book furnished by our Russian geological hosts. The upper section (A, B) is

Fig. 3.



Sections of the Baku Oil Field.

made along the axis of a pitching anticline, and the other two are transverse to the anticline, and about 1000 meters apart.

The dark seam represents the principal oil-bearing sandstone, which yields an oil having a specific gravity of 0.865 to 0.875. The lower beds yield salt water and some oil with a specific gravity of 0.885 to 0.920.

In the development of these oil-fields, the experience gained in our petroleum-regions has evidently been put to practical use; but apparently the mass of the people have but slight appreciation of what they owe to America in this respect. At least this was the impression I obtained from my personal observations, and from the way they received an admirable speech of Prof. I. C. White, of West Virginia, at the banquet given us at Balakhany, in which he sketched the development of the oil-industry in the world.

Their business methods are yet behind ours in many respects. Their oil, being heavier than ours, leaves a greater proportion of residues which are available as fuel. These are used for generating steam on their fleet of tank-steamers which carry the oil up the Volga far into the interior of Russia at a very low cost. They have pipe-lines from the wells to the refineries at the port, a distance of only a few miles; but the product is transported thence in tanks over the railroad which connects Baku with the Black Sea, and thence with the rest of Europe. As this road is owned by the Russian Government, and as it derives an important part of its revenue from transporting the oil, they have been unwilling to authorize the construction of a pipe-line to Batoum, the Black Sea port, which would so cheapen the oil delivered in Europe as to render it a formidable competitor to American oil. The Standard Oil people are watching developments here very closely; and I was told that the American Consul at Batoum, who receives no salary from our Government, is an oil-expert in their pay, and furnishes the most complete and reliable statistics extant with regard to the Baku oil-fields. The production of these fields in 1897 is said to have aggregated, in round numbers, 44,000,000 barrels, as against 63,000,000 for the whole United States. As oil-fields are already opened in other parts of Europe, largely in beds of similar age and constitution to those of Baku, it would seem that the time must be not far distant when America must yield the overwhelming supremacy it has hitherto held in the oil-trade. Here our greater enterprise, which has made not only our oil but our lamps a household necessity in the lowliest cabins of the remotest parts of the globe, is liable to prove a disadvantage, for our immense production has already begun to decrease, thus showing signs of an ultimate exhaustion of

our supposed boundless resources, while the slower-moving Europeans have only just commenced to develop theirs.

The theoretical question as to the origin of petroleum was one that naturally interested me. With us, I think, geologists are all agreed that it is organic, and only differ as to chemical method of its formation, and as to whether it comes from buried plant or animal life. In Europe, I was surprised to find that many geologists still held to the volcanic theory in one form or another, and still more that many of them regarded the mud-volcanoes, to which I have already alluded, as an evidence of this origin, though to me they are volcanic only in name, and are a result, rather than a cause, of the formation of petroleum. Our guide-book touched this question very lightly, simply saying: "The naphtha (natural) gases owe their origin and their activity to the mud-volcanoes, and to the salines of the peninsula."

In this connection, a paper read during the sessions of the Congress by A. Lebedintsev, a Russian chemical geologist, seems so pertinent that I give in the following paragraphs a brief summary of a part of it.

The Kara Bugas or Black Gulf is a shallow basin on the east side of the Caspian Sea, nearly opposite Baku, in the midst of a barren, lifeless desert which stretches far to the eastward. It has an area of 17,000 square kilometers, but is nowhere over 15 meters (50 feet) in depth, being connected with the Caspian by a narrow inlet less than 500 feet wide, and only 5 feet deep. The enormous evaporation from its surface in this dry climate causes a constant inflow of water from the Caspian, and results in some extremely interesting phenomena.

The enclosed basin of the Caspian receives an enormous amount of fresh water from the interior of Russia and the slopes of the Caucasus, on the north and west respectively, but very little from the desert country on the east and south; its waters have, therefore, very varying degrees of salinity in different parts. The salt of its waters differs very much from that of other seas, especially in its much greater proportion of sulphate of magnesia to chloride of sodium; this in the Black Sea, for instance, is only 1 to 11, whereas in the Caspian it is 1 to 2. Through constant evaporation this water in the Kara Bugas Gulf becomes greatly condensed, ranging in density from 1.3 Baumé

to 22–23 Baumé, the greatest density being naturally in the central and deepest part. Thus the percentage of common salt in the water rises to 12.8 per cent.; a double decomposition between NaCl and $MgSO_4$ takes place, and Glauber salt ($Na_2SO_4 + 10\ H_2O$) is formed, which may reach a proportion of 14 per cent. From the saturated solution at the bottom there forms a deposit; this is first, opposite the mouth of the inlet, largely gypsum, but in the middle of the basin it is Glauber salt in a layer of clear, limpid crystals, a foot thick in summer and probably thicker in winter. The area of this deposit is estimated at 3500 square kilometers, which, with an average thickness ot a foot, would mean 1,000,000,000 tons of Glauber salt. With cheap fuel near by at Baku, and water-communication already established throughout the Russian Empire, this would seem to furnish a basis for an important soda-manufacture.

The other interesting phenomenon, to which I have especially referred, is the following: The Caspian Sea is extremely rich in fish; the fisheries, we were told, yield an annual revenue to the Government of over half a million roubles. Shoals of these fish of various kinds make their way into the Kara Bugas Gulf, and, as the strong salt solution acts on them as a poison, they die and are washed out on the shores in immense numbers. The birds which congregate eat out their eyes and entrails, but do not touch their flesh, which, being thoroughly salted, is preserved from decay and serves as a food for the neighboring Turkoman tribes. It is easy to conceive how such masses of organic matter, covered after a little by sand and mud, and finally forming part of a buried sedimentary formation, might form an important source of petroleum, after the necessary chemical changes had taken place, and under favoring geological conditions.

The Black Sea.

The basin of the Black Sea presents phenomena of equal interest with those of the Caspian, and its surroundings are far more attractive. About six years since the Russian geologists obtained permission from their Government to make a study of the sea similar to that which has been carried on in the ocean by the British and American Governments in the "Challenger" and the "Blake" respectively. Before undertaking this work they had personally consulted Dr. John Murray, the geologist

of the "Challenger" expedition, as to methods of work, etc., and, as he was a member of our party, they made him the specially honored guest of the occasion, and put him in charge of the deep-sea dredgings and soundings that were conducted for our instruction and entertainment.

The basin of the Black Sea, as it will be seen from the accompanying copy (Fig. 4) of the bathymetric map given in our guide-book, has the shape of a somewhat curving ellipsoid, whose longer axis runs east and west. The two northern arms on either side of the Crimea, the Sea of Azof and the Gulf of Odessa, do not form part of the basin proper, being everywhere above the 100-fathom level. The sides of this basin are quite steep, especially on the north-eastern side, where, between Soukhoum-Kale and Novorossisk, the Caucasus mountains, coming to it at an angle from the south-east, are cut off abruptly at the coast. It would seem as if in some earlier geological period (probably at the close of the Jurassic), during an earth-movement, a part of the chain had been faulted down a few thousand feet. The south-eastern part of the Crimea presents a similar abruptly-cut mountain-face toward the sea, and in its geological structure resembles a segment cut out of the northern flanks of the Caucasus, the lowest beds exposed being Jurassic, cut through by laccolitic bodies of igneous rock, as at Kislowodsk, for instance. As seen from the sea, the present north-eastern coast, with its line of almost perpendicular bluffs, apparently quite straight and regular, shows evidence of a continuation of the movement along the fault-line into more recent (Post-Oligocene) times.

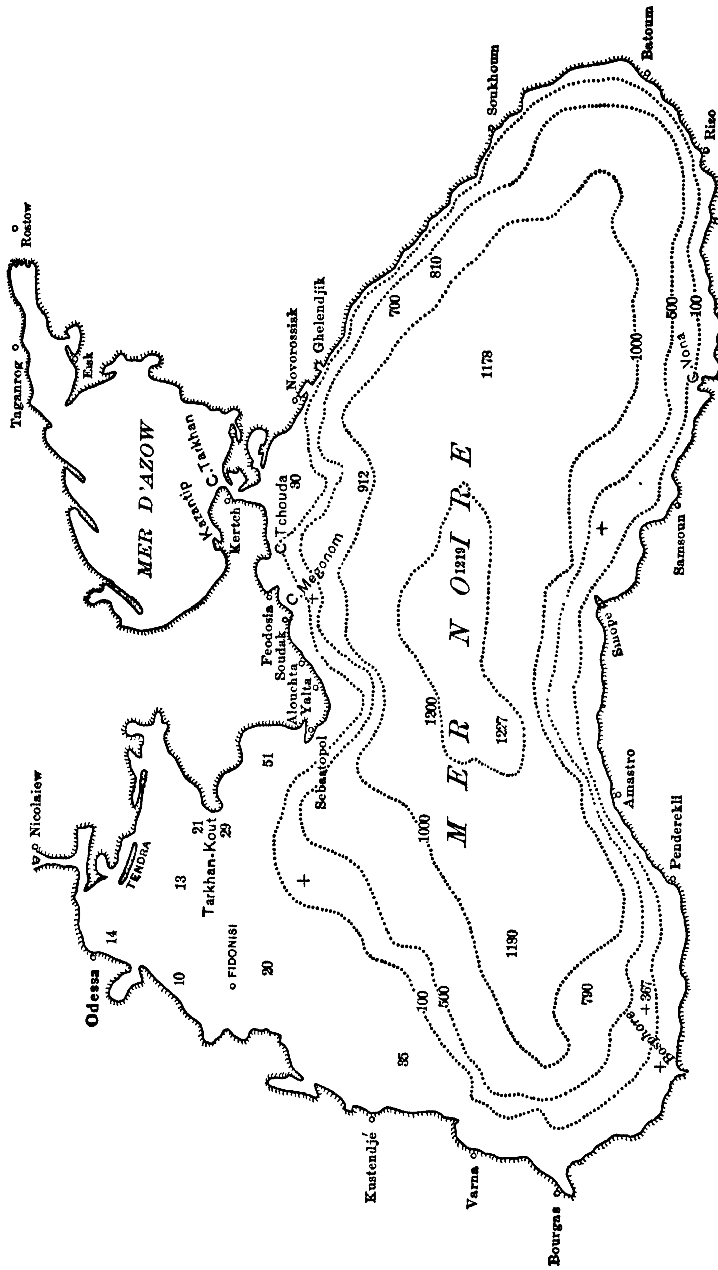
From the figures on the map, which give depth in fathoms, it will be seen that the greatest depths, in the center of the basin, are a little over 1200 fathoms (7200 feet).

The water of this basin presents four remarkable peculiarities:

1. *Temperature.*—This decreases downwards, at first very rapidly (summer observations), from 15° to 25° (Centigrade) near the surface to about 7° at depths of 30 to 45 fathoms. It then rises slowly to 8.8° at 100 fathoms, to 9° at 200 fathoms, and finally to 9.3° at 1200 fathoms.

2. *Salinity.*—Its salinity is everywhere less than that of the ocean. The average content of salt in its water at the surface

FIG. 4.



AMERICAN BANK NOTE CO. N.Y.

Bathymetric Map of the Black Sea.

and in the middle of the sea is 1.8 per cent. This increases to 2.1 per cent. at 100 fathoms and to 2.2 at 1200 fathoms.

3. *Sulphureted Hydrogen.*—From the depth of 100 fathoms

downwards the water contains an ever-increasing amount of H_2S. This ranges from 33 cubic centimeters per hundred liters (at 0° C. and atmospheric pressure) to 65 c.c. at 1185 fathoms.

4. *Absence of Life.*—As contrasted with the Caspian, where fish are found at all depths, fish-life is much less abundant in the Black Sea, and is entirely wanting below 100 fathoms.

The complete explanations of all these phenomena have not yet been worked out; but the line of reasoning thus far followed is very suggestive.

The Black Sea receives a great amount of fresh water from the many large rivers that empty into it on the north and west; the Don, the Dnieper and the Danube being the most important. This fresh water, on account of its inferior specific gravity, remains at the surface. The outlet of the Black Sea is through the narrow but deep channel of the Bosphorus into the Sea of Marmora, and thence through the Dardanelles into the Mediterranean. Now, in the Bosphorus there are two currents, running in opposite directions; a superficial outward current, which carries away the relatively fresh surface-waters of the Black Sea at a rate of 370,000 cubic feet per second; and a lower current, which brings in the heavier and more saline waters of the Mediterranean at the rate of 200,000 cubic feet per second. The waters of the latter current, which have in the Propontis 3.8 per cent. of salt, and upon their entrance into the Black Sea 3 per cent. of salt, mix but slightly with the fresh surface-waters, and in great part must sink to the bottom of the basin, carrying with them their relatively high temperature and greater degree of salinity. Now, the excess of water going out through the Bosphorus over that coming in is 170,000 cubic feet per second, or $\frac{1}{1700}$ of the entire volume of the Black Sea per annum. It would therefore take not less than 1700 years for the deep waters to be entirely renewed, while the surface-waters are constantly changing. The vertical currents are stopped at about the 100-fathom level by the denser layers below, and oxygen only reaches the latter by diffusion, and by the slow addition brought in with the lower currents from the Bosphorus, too slowly and in too small quantity, apparently, to sustain organic life, which is therefore confined to the upper layers.

The presence of H_2S in these waters is also unfavorable to organic life, the absence of which is thus not difficult to account

for; but it is more difficult to explain the abnormal contents of H_2S and of sulphides, especially of iron, which accompany it. According to some investigators, it is due to the activity of certain microbes which give out H_2S under the conditions that probably exist there. Andrussow attributes the origin of the H_2S, in part at least, to the decomposition of organic matter; the surface-waters abound in organisms (*Plankton*) which die and sink to the bottom, and, as there are no fish in the depths to feed on them, they must accumulate in great quantities and, by their decomposition, give rise to H_2S. I will not attempt here to enumerate the possible chemical processes that are supposed to result, as they have yet to be confirmed by the analyses of the materials from the depths which Mr. Lebedintsev is carrying on; but it is evident that they will be of great interest to the geologist as well as to the chemist.

Further interest attaches to the peculiar conditions of in- and out-flow of the Black Sea waters in view of Nansen's recent discovery that the region around the North Pole is largely an interior sea of a depth of 2000 fathoms or more.

Apparently this Arctic sea, though vastly greater, presents many analogies with the Black Sea. It has practically only one outlet—below a certain level, for the bottom of Behring Strait is above the level of the continental platform. Only between Greenland and Norway has it a deep-sea connection with the ocean, and through this it is receiving the denser and warmer waters which are brought up from equatorial regions by the Gulf Stream, and which probably sink in great measure to the bottom of the basin, while the surface-waters (which, being fed by all the north-flowing rivers of the northern hemisphere, are fresher and colder than those in the deeper part of the basin) are flowing out.

The Kotchkar Gold-Mines, Ural Mountains, Russia.

BY H. B. C. NITZE, BALTIMORE, MD., AND C. W. PURINGTON, BOSTON, MASS.*

(Atlantic City Meeting, February, 1898.)

THE Kotchkar mining-district, known as the Kotchkar System, is situated in the Orenburg government, in Eastern Russia, in the great plateau- or steppe-country immediately adjacent to the eastern slopes of the Ural mountains. Its longer dimension is north and south about 27 miles, while in the widest (northern) portion the distance across is 22 miles. The district lies 40 miles southwest of Miass, a station of the Trans-Siberian railway, and half-way between this village and that of Troitsk. It contains also the not inconsiderable settlements of Kotchkar and Kosobrodskoié, the last a Kossak town.

So far as explored, the district is known to contain from 300 to 400 auriferous veins.† Although the topography is generally of an exceedingly flat character, the country is traversed by several small rivulets running from west to east, and flowing into the Sanarka and Kamenka rivers. Since 1844 placers have been worked in the district, but it was not until 1867 that veins were discovered, and these have been worked only since the impoverishment of the alluvial deposits.

The deposit with which this paper especially deals is known as the Ouspensky mine, one of twelve owned by MM. de Zélenkoff & Cie., five of which are situated in the Kotchkar System. An opportunity of visiting this mine was afforded the writers, in connection with an excursion of the VII. session of the International Geological Congress, held in Russia during the summer of 1897. The mine is reached by a 40-mile "troika" journey from the railway station of Bichkil, itself

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* The writers wish to acknowledge their indebtedness to Mr. E. de Zélenkoff, of Kotchkar, who has tendered them much courteous assistance in the preparation of this paper

† "*Les gisements d' Or du systéme de Kotchkar, dans l' Oural du Sud,*" N. Wyssotsky, *Guide des Excursions du VII. Congiès Géologique Internationale*, 1897, St. Petersburg.

three days' journey by rail east from Moscow. The property is one of the oldest in the district, the concession having been granted in 1851. Its product, $2,735,000, forms a considerable percentage of that of the whole Kotchkar district, which has been, in round numbers, about $28,000,000. The Ouspensky shows more development than any other mine in the district, having five shafts, the deepest of which is 393 feet deep, and drifts to the total length of 1968 feet. It is probable that there are few mines in the Urals in which the amount of development exceeds the latter figure. As the Ouspensky is both geologically and economically typical of the Kotchkar deposits, a description of it may be regarded as applicable, in a general way, to the district as a whole.

The almost level plain is relieved, in the vicinity of the mines, only by the slight eminences afforded by the numerous heaps of tailings resulting from the working of the placers in former years. There are, besides, numerous small, shallow pits, evidently the efforts of prospectors. The surface-soil is of a rather poor character so far as agriculture is concerned, and can easily be referred to the disintegration of the granitic rocks by which the region is occupied. The rock itself lies but a few inches below the surface of the country, and in several places small outcrops are exposed. As there has been no movement by water of residual material, the placer-deposits, which were first worked in the district, consisted merely of the disintegrated rock and vein-stuff lying in place. It may be said here that this is often the character of the gold-placers adjacent to the Ural mountains. The word "saprolite," which Mr. G. F. Becker* has applied to similar occurrences in the southern United States, may well be used in describing this decomposed and untransported material. The country-rock is a fine-grained, gray granite, containing, besides quartz and orthoclase, biotite in considerable amount, with other constituents not easily determinable without microscopic examination. According to Mr. Wyssotsky, the rock forms part of a large zone which has a north and south direction. It has been subjected to decomposing influences to a remarkable degree.

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* "Gold-Fields of the Southern Appalachians," G. F. Becker, *XVI. Ann. Report U. S. Geol. Sur.*, 1895, part iii., p. 289.

In fact, certain portions have become so largely changed from their original character that a special name, that of beresite,* has been applied to the occurrences—somewhat superfluously, as it appears to the writers. The rock has been taken out, at a depth of over 100 feet, in a state apparently as soft and decomposed as at the surface.

The whole area in which the veins occur has been subjected to the action of deforming forces. These have produced, in the granite, zones of shearing, transforming the normal granite along certain parallel or approximately parallel lines into schist. The zones of schist are by no means regularly spaced, some occurring at intervals of a few feet, others being several hundred feet apart. Between the sheared zones the granite shows almost no indications of schistosity. The strike of the schistose belts is generally east and west, and since the gold-bearing veins occur along these, and since their direction has been determined by that of the schist, east and west is also the strike of the veins. Many vary in their direction by a few degrees to the northeast or the northwest. The dip of the schist and of the veins is steep. As noted in the Ouspensky mine, it varies from the vertical to 85° to the south. But in going toward the north, across the strike, it has been noted that the dip of the veins changes to the north. The exposures showing the schistose belts in section are, however, necessarily too limited to afford opportunity for any but the most general study. The region has not been sufficiently studied to determine whether or not faulting has played an important part. In the Mitrofanovsky property, however, two miles to the north of the Ouspensky, it has been found, according to M. Monchot,† that at the east end of the present workings a wall of granite, probably referable to a fault, completely cuts off the ore.

* The definition of beresite, as given in the paper of Mr. A. Karpinsky, descriptive of the region of Berezovsk, where it was originally found, is that of a rock consisting principally of quartz, muscovite and pyrite, approaching to greisen, and containing gold in greater or less quantity. It is not stated whether the gold is original or has been subsequently introduced. This definition would appear to apply to an occurrence somewhat different from that at Kotchkar. (See "*Versant Oriental de l' Oural d' Ourjon à Ekatherinebourg,*" by A. Karpinsky, in the *Guide des Excursions du VII. Congrès Géologique Internationale*, St. Petersburg, 1897.)

† *Résumé du Rapport du 27 Septembre* 1896, *sur les Mines appartenant a MM. de Zélenkoff & Cie.*

The veins, which contain much the larger share of the gold have been formed, it appears, most probably by the filling, through the agency of ore-bearing solutions, of the open spaces previously existing in the schistose zones. The physical appearance of the veins, as seen in the ore-breasts of the mine, bears evidence in favor of this hypothesis. The veins* are individually very narrow, but occur in belts of considerable width. The separate quartz-stringers average about one inch in thickness, and rarely exceed four inches. On the other hand, the whole belt of stringers, or what may, in a liberal sense, be termed the vein, averages about three feet in width, and occasionally reaches a thickness of more than twenty feet. The quartz stringers which contain the gold present all the characteristics common to true veins, the origin of which is subsequent to that of the rock in which they occur. Thus, in the faces exposed in the Ouspensky there occur in the quartz angular fragments of the granite sharply defined, whose origin must have been from the walls themselves. It is rarely the case that a single stringer extends for a great distance in any direction, but the lode has oftenest the character of a belt of stringers parallel, or approximately so, with intercalated bands of auriferous country-rock. The boundaries between the stringers and the country-rock are always well-defined, and this characteristic, together with the angular fragments noted above, precludes the hypothesis that molecular replacement took place in the formation of the ore.

The gold occurs, for the most part, in association with arsenical pyrites (mispickel), which is found in the veins of quartz, and also, to a small extent, impregnating the adjoining granite walls. Of the other metallic sulphides, pyrite is the most abundant, while small amounts of galena, chalcopyrite and stibnite also occur. As earthy gangue-minerals, it is probable that, beside the white and gray quartz, which is abundant, small quantities of chalcedony, calcite and chlorite occur. M. Boulangier,† in a report on the Kotchkar district, makes mention of antimony, and M. Monchot, in another re-

* Here the word "vein" is used in the strict sense defined originally by von Co ta.

† *Extraits d'un Rapport officiel addressé a M. le Ministre des Travaux Publics, le* 18 *Fevrier*, 1893, *sur les exploitations d' Or dans l' Oural, par M. Boulangier.*

port above noted, says that chlorite and tourmaline occur in the veins of the Mitrofanovsky. Tellurium occurs in the district, and in some veins is associated with the gold. Silver occurs with the gold in variable quantity, the highest proportion found being 30 per cent.

The ore, as mined and milled, consists for the most part of the quartz and auriferous sulphides of the vein, although the adjacent portions of the walls are at times sufficiently impregnated with auriferous pyrite to pay for mining and treating with the rest. Prof. F. Pošepný in his characteristically thorough account of the Ural deposits* does not accept the conclusions which have been arrived at by many geologists regarding the original character of the gold found finely disseminated in the igneous rocks of the Urals. He suggests, with much reason, that the most searching chemical and microscopic tests should be made before it can be declared with confidence that subsequent impregnation is not responsible for the presence of the gold.

The tenor of the ore in gold has been hitherto not constant. From 1872 to 1886 the principal vein of the Ouspensky property was worked continuously, and the ore ran more than 60 grammes to the ton, on an average—corresponding to $43.80. During these 15 years the mine produced over $1,200,000, although worked in a very small way. Below 150 feet, however, a zone of impoverishment was entered, which continued for 75 feet, where the ore has a value of from $8.00 to $10.00 to the ton. Below this a rich zone was again entered, but whether continuous for any distance is not known, as sufficient development work has not been done. The width of the main vein in the Ouspensky mine is also very variable. An average of the whole zone of stringers mined as ore is from 4 to 8 feet, with local widening to 20 feet. From comparative data, obtained by exploitation of several veins in the district, it is probable that the poor zone above referred to is present at about the same depth in all the veins of the region. Many properties in the district have been worked only to the depth at which the undecomposed sulphurets were encountered, since the operators possessed only such machinery as availed for the extrac-

* "*Die Golddistricte von Berezov und Mias am Ural,*" F. Pŏsepný, *Archiv für Praktische Geologie,* Vol. ii., 1895, p. 593.

tion of free gold. The milling and metallurgical machinery will be described below.

At the Ouspensky mine, as at all the mines in the district, horses are employed as the motive-power for raising the ore. The hoist is in the form of a large drum, with vertical axis, the horse moving in a circular track. Much work is performed by hand which, at places equally easy of access in the United States, would be done by machinery. But it must be remembered that the labor both of men and horses is exceedingly cheap, and if machinery is used, it must, for the most part, be imported, and transported for long distances. The cost of the native Tartar mining labor in the Kotchkar district is 60 kopecks (30 cents) a day, including board, which costs the company 4 roubles ($2.00) per month per man. At the Mitrofanovsky mine the cost of production is less than $5 to the ton of ore, exclusive of the cost of the chlorination of the concentrates, which amounts to $2.60 to $3.20 per ton of concentrates.

As the average tenor of the ore has been hitherto above $15 to the ton, it is evident that the enterprise is profitable. The item of expense which is probably the largest in proportion, is the cost of fuel, which is wood, and must often be transported for considerable distances. The cost at the mines is about $1 per cubic meter (about 76 cents per cubic yard, or $3.60 per cord). The character of the topography, however, renders the transporting of not only fuel, but all machinery and supplies, exceedingly easy and cheap. MM. de Zélenkoff & Cie. are replacing their steam- by gas-power, as they have determined that the cost of producing gas-power from coal imported from South Russia will be less than the present cost of producing steam-power with wood for fuel.

As pointed out by M. Boulangier,* it is very probable that there exist in the Urals a considerable number of auriferous deposits like those of Kotchkar, and that relatively few have been found. This opinion seems justified by the fact that even where very rich auriferous placers exist the source of the gold has rarely been discovered. At Berezovsk, in the vicinity of the city of Ekatherineburg, rich placers, which have yielded for many years, were visited by the writers. At that place,

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* *Op. cit.*

although gold-bearing veins have been found underlying the placers, there has as yet been little occasion to exploit them. At other places, however, as for example the Ilmen placer, near the town of Miass, on Lake Ilmen, no veins, so far as it was ascertained, had been found, although it is impossible not to think that thorough search would reveal them.

It seems less justifiable to suppose that placer-beds exist which have never been exploited; yet when one has seen the apparently unexplored forest-land of the Urals, it is not unreasonable to believe that new deposits may yet be found. Thus all indications lead to the opinion that the industry of quartz-mining in the Urals is but begun, that many opportunities for the establishment of enterprises exist in connection with long-known deposits, and that many more veins will be yet discovered. The facts that nearly all the old heaps of placer-tailings in the Kotchkar district are considered worthy of being re-worked by the cyanide or the chlorination process, and that already MM. de Zélenkoff & Cie. have extracted nearly $200,000 from the old tailings of the Ouspensky, at the rate of $16 to the ton, go to indicate the inefficient methods which have been used in the first washings of the auriferous material. It may not be out of place to add that the Russian mining laws are favorable to the granting of concessions to foreign investors.

It is only within the past few years that the milling and metallurgical processes have attained any degree of perfection in the Kotchkar mining district, and even to-day the loss of gold in the tailings must be considerable at some of the plants.

The type of crushing-machinery employed is universally that of the Chilean mill or arrastra, from the crudest to the most perfected design. Stamps have been tried in the district, but abandoned, owing to the excessive cost of shoes and dies, which it was necessary to import from Europe, as those made in Russia were of such inferior quality that it was impossible to use them.

For many years the extraction was limited to the amalgamation of the free gold on narrow plates of no great length, and the tailing-banks which have accumulated are consequently of such richness that they will, almost without exception, warrant re-working. Much has already been done towards obviating

these losses by the introduction of more perfect amalgamation-methods, and the subsequent chlorinating or cyaniding of the concentrates. But there is no doubt that some of the more refractory ores still present difficulties in their metallurgical treatment which, under the existing conditions, it will require further time and experimental study to overcome.

The most complete and successful plant in the district, at the time of our visit, was that of MM. de Zélenkoff & Cie., at the Ouspensky mine. The crushing-machinery consists of two Chilean mills, each having a capacity varying from 15 to 20 tons per 24 hours, according to the nature of the ore. Each mill is fitted with three iron rolls, having steel tires, and weighing about 3.5 tons each. These rolls are set at angles of 120°, and are connected by means of horizontal arms to the central vertical axis of the mill, making 8 revolutions per minute. The iron pan or mortar in which these rolls move is floored with steel, and is about 15 feet in diameter by 3 feet deep. The pulp discharges through a round-punched sheet-iron screen (⅛-inch holes), placed 1 foot above the floor of the pan, passing first over a stationary silvered copper amalgamating-plate, 12 feet in length; thence over a similar plate, 5 feet in length, having an end-percussion motion; and finally, by means of launders, also lined with silvered copper plates, to a series of Embrey concentrating-tables,* the feed troughs of which are lined with amalgamating-surfaces. It is thus obvious that every opportunity is given for the amalgamation of the free gold, and the loss from this source should be reduced to a minimum. In the treatment of the richer free-milling ores, mercury is also used directly in the pan of the mill.

There are 4 Embrey tables to each mill. The width of the smooth rubber belts is 4 feet, and the number of end-percussions per minute is about 220.

The entire plant is operated by a 35-horse-power engine. There is room in the building for a third mill, with 4 more Embrey tables.

The proportion of sulphurets (mostly arsenical) in the original ores treated varies from 2 to 5 per cent. The value of the

* M. de Zélenkoff owns the Russian patent-rights for the Embrey tables, which he builds at the very complete machine-shops situated at the Ouspensky mine.

concentrates varies from $100 to $750 per ton. The tailings from the Embrey tables run $3 per ton.

The concentrates are treated by the Deetken chlorination process, a modification of the old Plattner process. The plant near the Ouspensky mine is equipped with 6 double-shelf reverberatory roasting-furnaces and 12 chlorination-vats, with accessory precipitating-tanks, chlorine-generator, etc. The area of each roasting-hearth is 12 by 36 feet, the capacity of each furnace being 3 tons of concentrates per 24 hours, with a fuel-consumption of about 1 cubic meter (0.278 cord) of wood per ton of ore. Each furnace occupies 6 men per 24 hours, divided into 3 shifts of 2 men each. The ore is stirred with hand-rabbles, and salt is added to the charge during the last half-hour of the roast. The roasted ore is charged into the chlorinating-vats, where it is left 12 hours for chlorinating and 24 hours for leaching. The chlorine is generated by means of salt, manganese ore (of selected quality), and sulphuric acid. The filters, which are situated in false bottoms under the leaching-vats, are made of rags. The gold is precipitated from the chloride solutions by means of ferrous sulphate.

The extraction runs as high as 95 per cent. in the best adapted ores. Experimental chlorination-tests of galena- and tellurium-bearing ores yielded only 25 per cent. of the original assay-value, the loss being caused largely by volatilization during roasting.

Two men perform all the labor of chlorination, which is in operation only 12 hours of the day. The expense of treating 2½ to 3 tons of concentrates is stated to be 15 roubles (about $8).

MM. de Zélenkoff & Cie. have, at their Mitrofanovsky mine, a similar crushing-, amalgamation- and chlorination-plant, and also a small experimental cyanide-plant.

A newly organized French company, owning large concessions in the district, is erecting a cyanide-plant in Kotchkar, after the MacArthur-Forrest type, for treating old tailings, of which there are said to be some 2,000,000 tons in the various refuse dumps.

Mining Districts of Colombia.

BY HENRY G. GRANGER AND EDWARD B. TREVILLE, QUIBDO, COLOMBIA, S. A.

(Atlantic City Meeting, February, 1898.)

INTRODUCTORY.

THE Republic of Colombia is the northernmost country of South America. Its northern coast line extends from the frontier of Costa Rica to that of Venezuela, on the Caribbean Sea. On the west it fronts on the Pacific Ocean, along the Panamá Isthmus, and down to Ecuador. It is bounded on the south by Ecuador, Peru and Brazil, which latter country with Venezuela form its boundaries on the east. Its area exceeds 500,000 square miles.

It is deemed unnecessary to give here a map of the country. Fig. 1, a map of the department of Antioquia, studied in connection with a good modern general map of Colombia, will explain itself. Fig. 1 contains also some of the region of the Cauca valley, described in this paper.

Access.—The interior of Colombia is reached through the ports of Sabanilla and Cartagena on the Caribbean Sea, each of which places is connected with the Magdalena river by a short narrow-gauge railroad, running respectively to Barranquilla and Calamar. From these points lines of river-steamers serve the ports of the upper Magdalena, which they reach in journeys of a week to ten days. After leaving the steamer, a short railroad journey, followed by a long and tedious mule-ride, reaches the cities of Bogotá and Medellin.

Quibdo, at the head of the navigation of the Atrato, and its most important town, is connected by steamer-service with Cartagena, by canoe-routes with the various districts of the Choco, and by mule-road, at present well-nigh impassable, with the heart of Antioquia. On the western coast the port of Tumaco is a starting-point for canoe- and mule-travel to the southern part of the department of Cauca, while Buenaventura is the port of the central section of the valley of the Cauca,

which is reached by a short railroad and mule-road. This port is also a starting-point for the province of San Juan in the Choco.

The interior of the Darien is reached from either Colon or Panamá.

Transportation.—The transportation-facilities of Colombia are still in their infancy, and to this fact may undoubtedly be ascribed the relatively small development of its vast natural resources.

Saving the routes traversable by steamer or canoe, and the few short lines of railway now existing, all travel is on mule-back. Hence the weight of packages of merchandise or pieces of machinery is limited to about 150 pounds each. Up to 400 pounds may be carried in one piece by special freighters; and still heavier pieces have been carried on poles, by gangs of men, or pulled along the roads with block and tackle; but the cost of such transportation, except in rare instances, is prohibitive.

Many of the oldest roads were built up- and down-hill, after the system of the Indians. As Colombia is the most mountainous country on earth, this renders travel very difficult. In many of the worst parts of these roads side-hill connections on a contour are now being built.

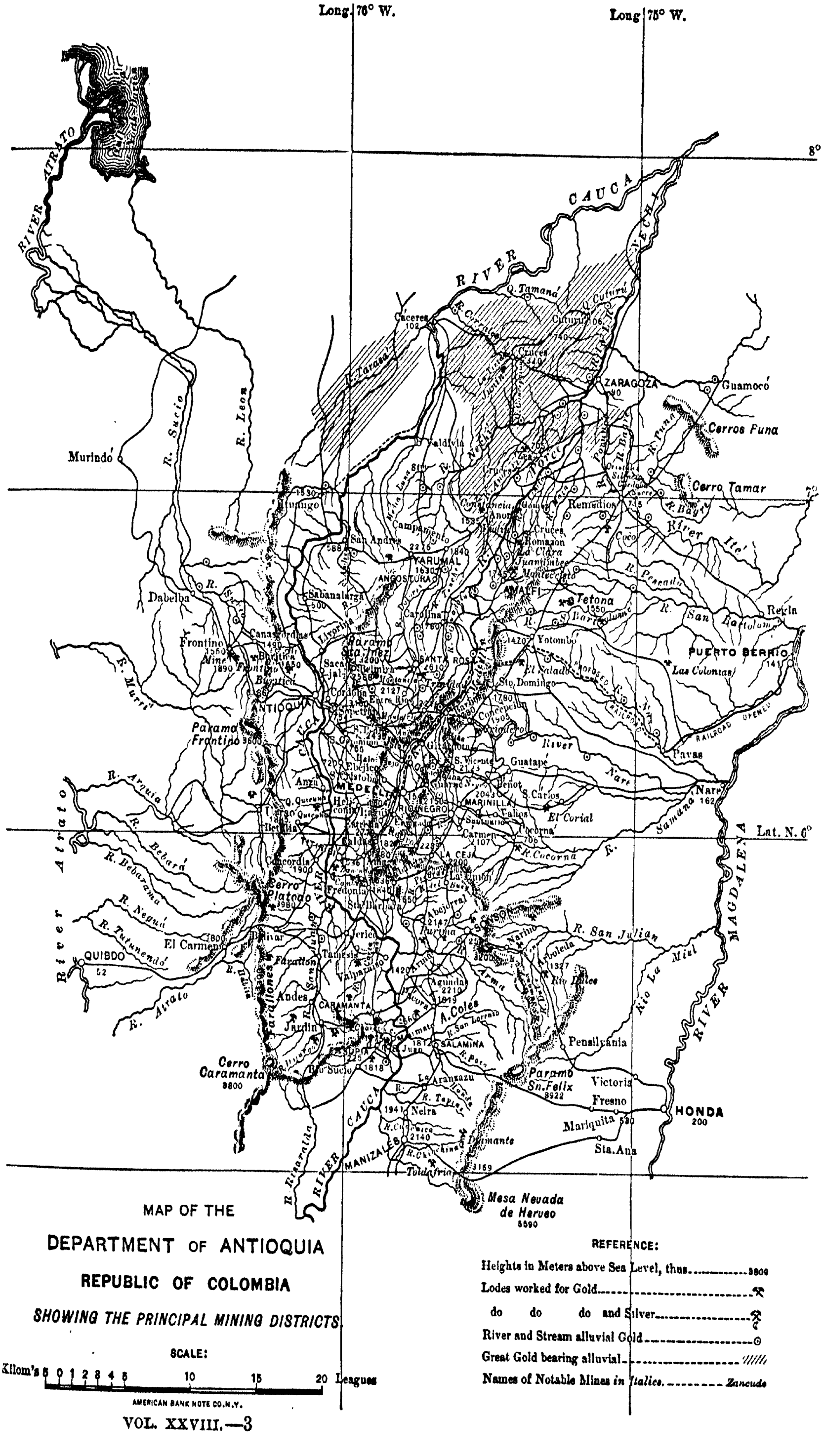
The interest manifested by the government and the people in the development of the short railway-lines now existing promises the eventual establishment of general railroad connections.

Climate.—Nearly every variety of climate, from equatorial heat to perpetual snows, is found in Colombia. The changes of altitude in a single day's journey are often such that the consequent climatic changes are apt to make the traveler very uncomfortable.

Some parts of the country, such as the Pacific coast, the Choco, and portions of the valley of the lower Magdalena, including the northern provinces of the department of Antioquia, are notably unhealthy, and should not be entered by the foreigner without due precaution. But the climate generally is healthy, and, in some parts, cool and invigorating.

Minerals.—A wide range of minerals is found in the varied geological formations of Colombia, although present exploitation is confined to but few.

Fig. 1.



Long. 76° W.
Long. 75° W.
8°
7°
Lat. N. 6°
RIVER ATRATO
RIVER CAUCA
RIVER NECHI
RIVER MAGDALENA
River Atrato
R. Sucio
R. Leon
R. Murri
R. Arquia
R. Bebara
R. Bebarama
R. Negua
R. Tutunendo
R. Atrato
River Nare
River Ité
R. San Julian
Rio La Miel
R. Samana
Murindó
Dabeiba
Frontino Mine
Paramo Frontino 3600
ANTIOQUIA
MEDELLIN
RIO NEGRO
MARINILLA
ZARAGOZA
Guamocó
Cerros Funa
Cerro Tamar
Remedios
Caceres 102
Yarumal
ANGOSTURA
AMALFI
Tetona
Yolombo
PUERTO BERRIO
Las Colonias
Pavas
Nare 162
Reria
Guatape
S. Carlos
El Corial
Cocorna
Cerro Platado 3980
Concordia 1900
Fredonia
Sta. Barbara
Abejorral
Sonson
Jerico
Tamesis
Valparaiso
Andes
CARAMANTA
Jardin
SUPIA
Rio Sucio 1818
Cerro Caramanta 3800
El Carmen
QUIBDO
A. Coles
SALAMINA
Aguadas
Pensilvania
Paramo Sn. Felix 3922
Victoria
Fresno
HONDA 200
Mariquita
Sta. Ana
Neira
MANIZALES
Mesa Nevada de Herveo 5590
MAP OF THE
DEPARTMENT OF ANTIOQUIA
REPUBLIC OF COLOMBIA
SHOWING THE PRINCIPAL MINING DISTRICTS
SCALE:
Kilom's 5 0 1 2 3 4 5 10 15 20 Leagues
AMERICAN BANK NOTE CO. N.Y.
REFERENCE:
Heights in Meters above Sea Level, thus 3800
Lodes worked for Gold
do do do and Silver
River and Stream alluvial Gold
Great Gold bearing alluvial
Names of Notable Mines in Italics. Zancudo

In the Department of Cundinamarca large deposits of copper are known to exist, but no modern attempt to work them has been made. A number of good prospects of lead-ores are likewise untouched; various deposits of cinnabar and antimony have been discovered in the cordilleras of the Cauca, but no exploitation has yet been attempted. Petroleum has been found in the northern section of the country, but of several desultory attempts to establish wells not one has resulted successfully, although many hold the opinion that work under a thoroughly experienced man would satisfactorily develop that important industry. Kaolin is exploited at Caldas, near Medellin, where a pottery has been established to meet the demands of the surrounding country.

Extensive beds of a very fair quality of bituminous coal are found in various sections of the country. It is mined at present in the sabana of Bogotá, where it is used for foundry-purposes, as also at Amaga and Titiribi in Antioquia. The only miner of the great deposits of the Cauca valley is a blacksmith at Cali, who periodically takes out a few barrelfuls for use in his forge. Concessions have been granted, at various times, on the coal-deposits existing on government land near Santa Marta and at other points of the northern part of the country, as well as on various beds of the Pacific coast; but such have been hitherto the limitations of these grants that capital cannot be induced to come in for development. Hematite iron-ore of fair quality is found at many places, and is worked, in a limited way only, for the use of the foundries of Bogotá and Amaga.

The only actual exploitation of manganese is on the Isthmus of Panamá, about 40 miles east of Colon, where an American company, organized by Sr. Eduardo J. Chibas in 1894, developed an important deposit, building for that purpose a 9-mile line of 36-inch-gauge railway. Of this important enterprise, now in successful operation, a recent paper in our *Transactions* gives an account.*

Emeralds are said to be found in various parts of the Republic. The famous mines of Muzo, near Bogotá, which were worked before the Conquest (the uncut gems supplying a large part of the booty of the sixteenth-century pirates), have been continuously worked under government-leases since the year

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* *Trans.*, xxvii., 63.

1568. No statistics of the output are available. The emeralds are found either in isolated crystals or together with uncrystallized corundum, imbedded in a mountain-side of limestone. The method of working is cutting away the barren surface-rock and then opening reservoirs to sluice off the débris until the limestone is laid bare, when the stones are broken out by expert laborers.

Large deposits of rock-salt are mined at Cipaquira, near Bogotá. The abundance of salt springs in Antioquia, Cauca and the Choco would indicate other bodies of this mineral. But all such deposits, and all springs whose waters exceed 6 per cent. in degree of saturation, are a monopoly of the government.

Among the minerals which are exploited to some extent in Colombia, silver and platinum play an important part; but that which has made the Republic famous as a mining country, and on the further development of which rests to a great extent its future prosperity, is pre-eminently gold.

Ancient Indian Gold.—When the Spaniards conquered South America, Colombia was a part of the immense tract then known as Peru; it was inhabited by a race and possessed a civilization akin to that of the Incas.

These harmless people were entirely helpless before the firearms of their Christian invaders; and the ashes of their beautiful cities yielded immense treasures of gold, often worked with consummate skill into figures relating to their religious worship, queer forms for personal adornment, or articles of practical utility in the household and in the chase. Even their fish-hooks were of gold.

History has left no record of the mining operations of these Indians, but it is probable that, according to their necessities, their supply of the precious metal was taken chiefly from the abundance of the gulches and creeks, where the Spaniards afterwards reaped such rich harvests with slave-labor.

Since the Conquest, systematic work has been carried on in Colombia, prospecting for and opening up the graves or "guacas" of the aborigines. Seldom is one found that does not yield rare pottery and stone implements, and a greater or smaller quantity of golden trinkets. It is by no means unusual that guacas in the Cauca, Antioquia or Choco yield from 1

to 3 pounds of gold, and some have been found which yielded as much as 25 pounds.

That the copper of the country was utilized by the Indians is shown by the fact that pieces of gold are occasionally found highly alloyed with copper.

Most of these specimens of ancient workmanship are conveyed to the melting-pot, but the interests of science are conserved by two valuable collections, one of the government at Bogotá, the other belonging to Sr. Arango of Medellin.

The Mining of the Spaniards.—The conquerors began mining about the year 1537, and the quantity of gold taken out since that time places Colombia third amongst the gold-producing countries. The work was done by forcing the labor of the Indians, and by the introduction of negro slaves.

The Spanish taskmasters were merciless, and drove the Indians far beyond their powers of endurance, with the result of great mortality amongst them. Stories are told of the terrible vengeance wreaked by these natives when opportunity offered; how they would capture the hated driver and carry him off to the mountain fastnesses known only to themselves; how they would tie their oppressor to a tree, and, forcing open his mouth, give him his fill of gold, molten hot. These weird tales have been corroborated by the subsequent finding of skeletons bound to trees with a lump of gold at their base. Later, through the intervention of the missionaries, laws were passed prohibiting the enslavement of the Indians; but the legends amongst them of the cruelties endured by their forefathers for the sake of gold have caused them to abhor the yellow metal, and now they neither mine nor use it in any form.

The employment of negro slaves continued until their liberation, some fifty years ago.

About the middle of the sixteenth century slave-workings were started in the placer-districts of the northern part of Antioquia and the south of Cauca, and the exploitation of these gravels continues until the present day. Towards the latter part of the same century the first vein-mines were opened up in Antioquia.

The mode of working was extremely crude, the ore being mined out with picks and wedges and crushed by rude dolleys.

Such work could not have been done, save for the fact that the soil yielded the necessary supplies with but little labor, thus making all returns, however low, almost wholly net profit. In the year 1600 work was begun on the placers of the rivers flowing from the western Cordillera to the Pacific, and was continued by the "Haciendas" until the liberation of the slaves; and up to the present time the negroes, who are almost the only inhabitants of these rivers, still "batea" out sufficient to provide for their few wants. These rivers include the Anchicaya, Cajambre, Timbiqui, Naya Guapi, Guajui, and others. On an exploration-trip which we made last year to some of these, we were astonished at the great bodies of gravel, aggregating many millions of cubic yards, which had been moved entirely by hand-labor, assisted in some instances by water brought in ditches of sometimes more than a league in length, and in others only by the rain-water that could be caught by small dams.

The effect of such workings is seen at the mouths of the rivers, which are, as a rule, quite shallow, and have great sand-bars projecting out into the sea.

The washings were evidently first started in the concentrated riches of the creek-beds and gulches, and then on the recent gravel benches left by the rivers when, changing their course, they cut newer and deeper channels. At one of these old workings on the Rio Sucio, a tributary of the Cajambre, the objective point was a basin, to drain which the bed-rock was channeled several hundred yards to an average depth of about 20 feet. The bed-rock was a hard slate; and as all the work was done by hand, it must have proved a wearisome task.

Similar is the history of the Spanish workings in the Choco. The slaves were worked in large gangs, and took out fortunes for their masters until the day of their liberation; after which the whites nearly all moved to the healthy regions of the interior, leaving the negro, with his batea, in full possession of the ground; but such is the natural indolence of the black, that he works the mines only enough to supply his actual wants, which are very few.

The Crown aided the mining industry by sending out from time to time Spaniards who had gained experience in the mines of Germany, and in some instances German engineers, who

brought about gradual improvements in the methods of work. The Crown attempted to work some of the vein-mines of the interior on its own account; but, as generally happens when a government undertakes enterprises outside of its legitimate business, the expenses were greater than the returns. Private capital subsequently carried on the work with better success.

In the early days the tax was very heavy, being 20 per cent. on the output of both gold and silver, but this was gradually reduced until, shortly after the establishment of the Republic, the example of progressive countries was followed, and the production of precious metals was made free.

Gold-Production.—Authorities differ greatly as to the quantity of gold exported from Colombia. There are no means of reaching an accurate knowledge of the output, but an estimate, said to be conservative, gives it (in American money) as follows:

Antioquia,	$226,800,000
Cauca, outside of Choco,	96,000,000
Choco,	126,000,000
Panamá,	67,000,000
Tolima,	36,000,000
Santander,	11,500,000
Bolivar,	5,500,000
Cundinamarca,	2,000,000
Boyaca and Magdalena,	400,000
Total,	$571,200,000

These figures are to the end of the year 1875, since which time some $55,000,000 has been taken out, making the total product to date $626,000,000.

The silver output to date is estimated at $61,000,000.

The platinum output to date is estimated at 18,000 kilogrammes.

Mining on the Isthmus.

The first gold mined by the Spaniards in Colombia was discovered by Columbus on the Isthmus of Panamá on his fourth voyage, in the year 1503, and the great navigator and his crew were the first Europeans to wash the gold-sands of the Americas.

The Darien streams are all more or less auriferous, and washing their sands has continued until the present day.

Mr. C. A. Barnes, an American, established gravel-workings near Panamá, with successful results, in a small way. He

died in 1895, during a journey of exploration to the Choco. The most widely-known mine of this department is the Espiritu Santo, at Cuna, a quartz-mine opened up in 1680, and abandoned in 1727 on account of the attacks of the Cuna Indians. This mine undoubtedly produced great quantities of gold, although many of the accounts given of its ancient richness are probably exaggerated. In 1884 it was rediscovered and reopened by the Darien Gold Mining Co., and under the able management of Mr. Thomas H. Leggett, now prominent in South African mining circles, produced for a time successful results; but later the veins became pockety, and the output unsatisfactory.

There are numerous known veins on the isthmus, and probably many more to be found. From one claim, located on the Pacific side, specimens were sent to us assaying an average of $12.00 per ton in free gold. The mine was said to be in favorable condition for working, but other engagements prevented our visiting it.

Mines in the Valley of the Cauca.

The first work in this department was done on the gravels near Popayan, where desultory washing is still kept up by the negroes; but our comments will be limited to those mines at which work on a regular scale is now going on, or has been recently attempted.

Chantaduro.—This property lies on the eastern slope of the western Cordillera, about 20 miles south of Cali, and comprises over 1000 acres. The greater part of the surface is covered to a depth of from 6 to 10 feet with a deposit of ancient gravel of a generally fine character, but with a liberal supply of immense glacial boulders scattered through it. The value is stated at from 20 to 40 cents per cubic yard. The bed-rock is a fairly tough clay, showing occasional thread-like deposits of hematite ore. The property was purchased in 1887 by an American company, but serious errors in judgment were committed by its manager. Instead of bringing a water-supply from the river Jamundi, a couple of leagues away, he erected dams to catch rain-water, and without any distinct detailed plans he ordered out some two miles of piping and a couple of large-sized "giants." Work was moderately successful during

the rainy season, but had to be suspended for lack of water during the long dry spells; and for this and other reasons the manager was removed. His successor reported so unfavorably on everything that had been done that the stockholders were discouraged, and shut down the work. Later, Mr. John W. Pace, of California, leased the property and worked it with fair results during the rainy weather with his partner, Mr. James Orr, until the latter was drowned in attempting to cross a swollen stream at night. Since then the property has been abandoned to the negroes.

Peon and Jardin.—The Peon and Jardin adjoins the Chantaduro on the north, and its general character is similar, with the advantage that it is considerably nearer to water-supply. The attempt to work this property was unfortunately typical in many ways of methods only too common. Several Americans came to the valley in 1894, and with the assistance of an Englishman resident in Cali, secured a long option on the property, agreeing to give him a certain percentage of the profit of any sale that should be made. This, he asserts, they never did, though the property was unloaded upon a Scotch company at a large profit, and upon the strength of extravagant representations, supported by the report of the agent (subsequently superintendent) for the purchasers. The managers sent out by this company were without mining experience; and the observations made by one of the writers, who visited the place in 1895, led him to feel grave doubts concerning the success of the enterprise. Between the mines and the terminus of the Buenaventura railroad there are some 73 miles of mule-road. Notwithstanding this, the pipe for the work was ordered ready riveted in 3-foot sections, and nothing lighter than No. 10 iron. The grade for the ditch was determined by aneroid barometer-readings, and upon this basis the point of supply at the stream was fixed, and a large gang of men was set to work opening the ditch without any preliminary survey or study of the nature of the ground. After digging a large section they ran into solid rock at an elevation where lumber for flumes would have cost a fortune. The working-capital was exhausted in blasting, and a halt was called.

An English engineer named King was then sent out to report. We understand he advised that, since so much had been

spent, it would be well to go a little further and make practical tests. For this purpose an American, Mr. Holland, was sent. He reported that no arrangements could be made for dumping-facilities, and advised a shut-down, which accordingly took place. The large supply of boiler-iron sections still lies at the terminus of the railroad, but little having been transported to the mine.

The business way of handling this affair would have been to prove the gravels thoroughly, secure all necessary dump-privileges, and ascertain the practicable course of the ditch, in addition to thoroughly proving the gravels, before spending a cent in purchase or other work. These points having been satisfactorily settled, common sense should have shown that for such low banks, on such steep slopes, a maximum pressure of 150 or even 100 feet would have been ample, and this could have been attained with but a few hundred feet of piping, the water being brought from the ditch in wooden flumes. For such a fall a small diameter would give a large flow, and No. 16 iron be amply strong. A few mule-cargoes of well-nested pipe and a couple of No. 1 monitors, which, with the light pipe, could be quickly moved and reset, aided by water from the ditch for a ground-sluice, would have handled the material very rapidly, working around the boulders. By means of close-following boxes, which would avoid puddling the bed-rock, and in which no attempt need be made to save gold, the cargo could have been run over a series of grizzlies, and then the gold saved in a permanent line of sluices with undercurrents.

Garcia Mine.—We have not visited the Garcia mine, about 30 miles south of Cali, on the western slope of the central Cordillera. Competent mining men who have done so inform us that it is one of the best-situated in the Cauca valley, the property including both water and dump, and the gravels being washable at a profit. Unfortunately, the owners of this property employed as superintendent a rather erratic character, whose misuse of the funds entrusted to him for the purchase of equipment was ascertained by them before the complete establishment of workings, and caused the abandonment of developments.

Socorro Mine.—This mine is near the summit of a high mountain in the western Cordillera, near Cali; the work con-

sists of ground-sluicing into short boxes the gravels of the bed of the *quebrada* (creek) Socorro. It is the property of a local company. Developments were started during the latter part of 1894, under English management, and the products covered the expenses for a year. Later, under the management of Mr. Pace, more favorable results were obtained.

Ensolvado Mine.—This lies some 35 miles south of Cali. A local company operated a small wooden stamp-mill on the quartz veins of this property, for a time, with fair results; but for the past few years it has been tied up in litigation, which has kept it idle.

La Maria.—This is a quartz-property, near Buga, on the west side of the central range, owned by a company of wealthy coffee-planters. Nobody seems to have known anything about the mine except the superintendent, who reported having struck ounce-ore in a narrow vein. He died; the mine caved in, and has lain idle for a number of years.

La Para Mountain.—Four days' journey after leaving the beautiful valley of the Cauca, one reaches the town of Supia, in the mountain regions on the west side of the Cauca river (see map, Fig. 1). Although Supia lies at an elevation of some 4000 feet, it is so entirely shut in by mountains as to be hot and unhealthy. After recovering from a few days of fever, we rode up the Para or Turkey peak, which rises east of the town.

The mines are all on the western slope.

Although the outcrops of various veins are clearly to be seen, only one has been developed. In this, six claims have been located, called, in order of ascent, Guadualito, Mercedes, Trinidad Primera Zona, Trinidad Segunda Zona, Platanar, and Libia Vieja. These are owned by separate companies, but the ruling figure in each is Don José Jesus Hernandez, whose various mining interests have made him the second wealthiest man in Colombia.

The ore is silver glance, yielding (when sorted) from 60 to 1400 ounces silver per ton, with but a trace of gold. The vein is 2 feet 10 inches wide, vertical, and strikes due E. and W.; it is a contact-vein, the north wall being blue porphyry, while the south wall of the first three claims is blue porphyry, which, further on, breaks into *arenisca* or sandstone. The country

on both sides shows more or less mineralization, and the entire quartz matrix carries a sufficient quantity of metal to be a very important factor, were it concentrated and treated by a modern continuous process. The ore is mined by over-hand stoping and brought out in wheelbarrows to sheds at the mouths of the various adit-levels, where it is picked over by women, who send to the mills only what their experience tells them will yield at least 60 ounces per ton. The remainder goes to the dump.

There are fifty hands (men, women and children) at the mines. The men receive $1.00 paper or about 40 cents gold per day, and the others proportionally less. Work begins at 6 A.M. and lasts till 5 P.M. The product varies from 12 to 30 arrobas (arroba = 25 pounds) of bar-silver per month.

The mineral from La Para is freighted on ox-back, at a cost of $1.75 gold per ton, to the amalgamating plant "La Amalia," about 5 miles to the N. W., on the northern side of the basin of Supia.

The plant is under the charge of Don Pablo E. Mejia, whose competent management has greatly improved the results in extraction.

Our visit took place in the dry season, the supply of water being reduced to 0.6 cubic feet per second. This was used to run an over-shot wheel, 22 feet in diameter and 2 feet wide, which was connected with a wooden dry-crushing battery of 8 stamps. Owing to low water only 4 stamps were on duty, dropping 18 times per minute. The ore is stamped to pass a 30-mesh sieve, the product being sifted by the battery-feeder. With plenty of water, as in the rainy season, the stamps drop 30 times per minute, and are claimed to crush 500 pounds per head every 24 hours.

The mill-man gets 32 cents gold per day when only 4 stamps are dropping, and 40 cents when all 8 are in use. Shifts are 12 hours.

The roasting-furnace, a double reverberatory of stone, has a uniform width of 12 feet and height of 6 feet. One side is 18 feet long, from the chimney (which is 45 feet high) to the end; the other is 16 feet long. It is fired from the end and loaded from the top. The arch is 18 inches high in the center. The ore is all worked from one side and discharged into cooling-bins on the other. The furnace-men are so expert as to tell,

by the feel of the ore on their rods, within a few ounces of what the yield will be per charge.

These men work in 24-hour shifts, earning 56 cents gold per shift, and feeding themselves; they then "lay off" for a like period, and "paint the town red." The furnace has a capacity of 210 arrobas, or 2.6 tons, per 24 hours. Ten mule-loads of wood, costing $2.36 gold, supply fuel for this period.

Each charge for each single furnace is 35 arrobas of ore and 1.75 arroba of salt, more or less, according to the estimated contents in silver. Of this salt 1.25 arrobas are from neighboring salt-springs and 0.5 arroba is sea salt, which is supposed to add some special virtue.

The ore is gently calcined for 6 hours, and then salted and roasted for 1.5 or 2 hours more, thus allowing three charges to be worked in 24 hours. No charcoal is used with the ore.

After the sulphur, arsenic, etc., have been driven off, the roasted ore is taken to the adjoining amalgamation-building and there treated by the barrel-system. The tail-race from the battery-wheel runs a second wheel of like dimensions, set on a long wooden shaft. On each side, close to the wheel, is a 5-foot wooden spur-wheel, which gears with similar wheels set on the end of two barrels, 5 feet in diameter and about 8 feet long. By a system of levers any one of the four barrels may be thrown out of gear at will. Each barrel is loaded with half a ton of roasted ore and about 1 cwt. of chunks of iron. The barrel is run 3 hours before adding mercury (200 pounds) and then run 18 hours before drawing. The speed varies from 14 to 18 revolutions per minute. The average product is 30 pounds of amalgam, which, retorted and melted, gives 16 to 17 per cent. of its weight in bars .950 fine or better. They carry 3 grains of gold per pound. The amalgam is run into troughs leading to the bag-tray, and the charge is run out over a set of sluices with deep riffles on a steep grade, and then a longer box, at very low grade, followed by another set of deep riffles, thus catching all the amalgam. The charge is next run into a large settling-box, from which it passes on to the pile of tailings, after having been partially concentrated by shoveling. When ore is scarce, the tailings are re-worked through the whole process, yielding about 10 per cent. of the yield of fresh ore.

An approximate estimate of the cost per ton would be as follows:

	American gold.
Mining (including hand-breaking and picking),	$7.50
Freight,	1.75
Salt,	4.85
Fuel,	.90
Labor at furnace,	.40
" amalgamating,	.40
General expenses, wear and tear, etc.,	3.00
Total,	$18.80

Assuming that the loss in weight by roasting is 40 per cent., the yield per ton would be 109 ounces. So silver will have to drop still lower before this mine has to shut down.

Were this vein near a railroad, it would doubtless be fitted up to pay handsomely. One attempt was made to equip it with a pan made by an unknown American foundry, which proved a flat failure both as a grinder and as an amalgamator; and the experiment was so costly that it is not likely to be repeated in the near future.

Marmato.—Traveling east from La Para for a couple of hours, we reached a town, the houses of which are built on steps cut in the steep rocky slope of the Cordillera of the western bank of the Cauca, and under which run the many drifts of the Marmato mines, owned by the Western Andes Mining Co., Ltd., of London. Fig. 2 is a view of the town from the plaza.

This property, which includes some 2000 hectares (4940 acres), has been worked for 357 years, having been first opened in 1539. Although, in this long period, the enormous network of veins whose outcrops are everywhere visible has scarcely been touched, still the present developments would require for inspection many times more than the single day we devoted to them. Our desire to see other mines further on was somewhat augmented by the fact that one-third of the population of this place was down with small-pox, and another third just recovering. Contrary to the sanitary regulations of most countries, and indeed of the greater part of Colombia as well, no measures are taken in this section to indicate the houses of the afflicted or to prevent the spread of the plague, the not unfre-

quent visits of which are simply regarded as an expression of the wrath of Providence.

However, three hours underground with Capt. Charles E. Dabb, a rapid survey of the plant, and some hasty notes, supplemented by those given us by Mr. Charles W. Brandon, the chief owner and general superintendent, will enable us to give some idea of this noteworthy property.

The altitudes above sea-level of the river Cauca, of the office (a little above the mean of the different adit-levels), and of the top of the mountain, are respectively 2359, 4550 and 5497 feet. Fig. 3 is a view of the mountain. The country-rock is a hard porphyry. About a mile south of the town the company is working a silver-vein to which little attention is given, and the output is but a ton or two per day, averaging 100 ounces per ton. The treatment is similar to the method employed at La Amalia.

According to Mr. Brandon, when operations were started on this deposit in 1814, they worked out in 8 years a cap covering some 5 acres, which ran as high as 5000 ounces per ton, the ore being ruby-silver. There are several other silver-veins known, one of them being leased at a gross royalty of 25 per cent. to local parties, who are fitting up a small plant to work it; but the great majority of the Marmato quartz-veins carry exclusively gold, the metal being in compact sulphurets. Operations are running on six main lodes, closely parallel, and each having a solid face from 2 to 9 feet in width, sometimes even wider. The owners claim that they can run a cross-drive a mile long and strike a workable vein every 6 or 8 fathoms, and the outcrops seem to bear out this statement. The general trend of the veins is from N.N.W. to E.S.E., and the dip is vertical, with occasional slight variations to the south, and sometimes as far as 65° to the north. The walls are very firm and little timbering is required—a great advantage in view of the scarcity of suitable timber in that region. All work is over-hand stoping. From the several adits, at different levels, the ore is run to the mills in tram-cars, pushed by women, on 10-pound rails. In the levels, one is often knee-deep in water, there being no drainage-gutter, and some of the former engineers having been very careless as to grade.

Another matter which Capt. Dabb is attempting to reform is

the manner of loading the ores on the cars. In place of having chutes from the stopes to empty into the trams, the present mode is simply wheeling the ore to the end of the stope-level and dumping it on the track, whence it is shoveled into the cars. This makes passage quite difficult and somewhat dangerous.

The full complement of labor in the mines and mills is 350 men and 50 women. Miners receive 56 to 64 cents (gold) per day, helpers 24 to 32 cents, and women 20 cents. All feed themselves.

The output is from 1000 to 2000 tons per month, and the estimated cost of mining alone is $1.20 to $1.60 (gold) per ton. The assay-value varies from 12 pennyweights to 1.5 ounces per ton. The extraction is extremely low, often amounting to no more than 4 pennyweights. The total product is from 400 to 500 ounces of gold per month.

Milling is done in a string of 11 mills, attended by 1 or 2 mill-men each. The largest of these have batteries of 15 stamps, in sets of 5; the total dropping is 147 heads. They are operated by over-shot water-wheels, the same water running the whole string. The difference in level from the upper to the lowest mill is 700 feet. Fig. 4 shows the batteries of No. 3, a 15-stamp-mill.

At the time of our visit the water-supply was 4 cubic feet per second, and the stamps were making 36 drops per minute. In the rainy season the speed runs up as high as 70; in very dry weather as low as 15. The height of the drop is 12 inches when the wooden cams are new.

The mortars are loaded full and left until the signal-pin sounds for more ore; the dies are stamped ore or porphyry; the wooden stems weigh 215 pounds, and the chilled-steel heads, when new, 160 pounds, making the total weight of stamp 375 pounds.

The crushing is very coarse, 11-mesh screens being used. Two of the mills have arrastras to grind the concentrates from the blankets and from one buddling-table.

Mr. Robert B. Johnston, who was in charge of the work of erecting a plant for the treatment of the tailings by the McArthur–Forrest process, claimed that he would be able to extract 75 per cent. of the contents at a cost of $3 per ton.

We have learned since that, owing to disagreements between the stock- and bond-holders, this work has been suspended.

Although the company loses so much of the contents of its ores, this is not wholly a dead loss to the world. One of the sights that first call the visitor's attention is the number of green beds, about the size of flower-beds, which, with their lower side walled up to a level, appear by the houses of nearly all the peon families. These are what they call their "Jardin-de-oro," or gold-gardens. Fig. 5 gives a view of them.

The wives and children of the laborers may be seen busy at work along the precipitous course of the tail-race making *batea*-concentrations of the sulphurets, which they carry off to their gardens to decompose. There they are periodically panned over, and new collections are added to what may remain incompletely oxidized.

On the Marmato mine the gold is all saved on blankets, it being claimed that it is rust-coated and will not amalgamate. It would seem that this difficulty might be overcome by fine crushing, but desirable reforms will probably be long delayed, owing to the entanglement of the company in lawsuits.

The lodes of this portion of the property are true fissure-veins, and are traceable all the way down to the Cauca river.

The property, despite the high freight-charges for machinery, certainly appears to warrant the establishment of a modern plant to treat the ores with fine crushing, close concentration and the cyanide process; the mill to be situated at the water's edge. The water in actual use would supply hydraulic force for all possible requirements.

The use of power-drills, attacking the mine from the river-level, in connection with a suitable plant, would make this property a handsome producer, and one that would have ore left for future generations.

Echandia.—In plain sight of Marmato, in a northerly direction, near the summit of the Cerro de Loaiza, lies the town of Echandia, which is reached in a half-hour's ride up the mountain-side. Here are the mines which laid the foundation of the fortune of Don Bartolome Chaves, the richest man in Colombia.

These mines, although their returns are far surpassed by a number of others, have caused great excitement at various

times by reason of the striking of rich pockets, where gold has been broken off in pound-chunks. The stories resulting have gathered increment in proportion to the distance from the field. In some of the islands of the West Indies it is believed that Chaves's mine (referring to Echandia, although he has many others) is guarded by a strongly-locked iron door, and that the owner is the only one who ever enters, and he only when he wants to chisel out a few pounds for current expenses. This is but one of the absurdities we heard or saw in print long before we had the pleasure of meeting Sr. Chaves at Supia, and his manager, Don Ricardo Eastman, at Echandia.

The property of Echandia includes the main mine and mill, on the southeast side of the Cerro; the Chaburquia mine and a small mill, on the north; and the amalgamating-plant at La Linea, 7 miles back on the range, besides a large tract of timber-land.

Since opening up in 1867 nearly $3,000,000 (American gold) value of bullion has been produced.

At the time of our visit little work was going on, the dry weather having cut off the water-supply; but Sr. Eastman and Don Guillermo Escobar, the engineer, escorted us through such of the drifts as were kept open, and assisted us in gathering some data.

The lowest levels are 5200 feet above the sea. The veins on both sides of the Cerro have the same course, N.N.W.–S.S.E., and dips varying from vertical to 50° N. The country-rock on both sides is a hard, blue porphyry, called by the miners *ojo de muerto*, or "dead man's eye," because of the large size and dead-white color of the crystals. The formation is cut by a dike of diorite, which has been reached at a distance of some 1200 meters by several galleries on the south side. On the Echandia, or south, side the mineral has been nearly all worked out above the level of the mill, which is some 300 feet below the top of the Cerro; but the veins all appear to go down in good form, and it is intended to attack them at a lower level as soon as the required rights can be obtained from the Marmato Co., whose lands are adjoining.

One vein, called the "Macha," which is untouched because too poor to pay the cost under present methods, stands 18 feet wide and carries 40 ounces of silver and 4 pennyweights of

gold per ton. The veins mined run from 1 to 5 feet wide. The gangue is a soft *caliche*, or amorphous carbonate of lime, and the matrices are quartz and feldspar. These veins occasionally pinch to as little as 1.5 inch and widen to over 10 feet, and the value of their contents varies inversely with the width. They carry iron pyrites, galena, zinc-blende, arsenides, copper pyrites, native silver and gold. The average value per ton is $30 (American gold), 90 per cent. of which is silver, and the remaining 10 per cent. gold.

In 1870 a rich *clavo* was discovered in the vein "Bocajoyo," which yielded about $75,000 in chunks of gold encrusted with quartz, one piece weighing 22 pounds.

At the time of our visit to Echandia the only work going on was that of a few "contractors," who mine out the ore wherever they can find it, making no effort to keep up galleries, while their wives and children carry the ore to the tram (which runs on strap-rails) and wheel it out. Naturally, this gives a sort of rabbit-burrow appearance to the workings.

Outside, a few men and women were also at work, grinding on stones, by hand, the richer parts of the outcrop. They pay 25 per cent. of the free gold as a royalty. The gold they extract is perfectly black, and the only indication of its precious nature is its weight. Its value is only about $7 per ounce.

The Echandia mill is equipped with 3 wooden batteries, with a total of 27 stamps, 5 hand-jigs, which are used to free the ore from the *caliche*, and a *gandinguero*, or buddling-table, which effects a 30 per cent. concentration. This concentrated ore is treated by the plant at La Linea, which cost $50,000 (American gold), and is said to have been the first of its kind in Colombia. It consists of 4 furnaces, 10 barrels, 2 8-stamp batteries, a laboratory, and brick- and tile-works. Its maximum capacity is 16 tons per day of 24 hours.

Chaburquia.—The Chaburquia mine has been working since 1869, at an average output of 40 tons per month, on the system of purchasing from the miners the clean ores at the rate of 60 cents per cargo of 250 pounds. This has resulted in a deplorable condition of workings. In some places we could pass only by forcing ourselves sidewise between the walls of the galleries. The miners follow a rich streak down as far as their notched-pole ladders will reach, carrying out the seepage-water

in calabashes and the ore in hide-sacks. However, they are now running an adit to cut the veins at a lower level, and bottom their contents in a workmanlike manner. The two principal veins are about 2 feet wide. They carry an average value of 60 ounces silver and 3 pennyweights gold per ton.

A 5-foot vein of *negro-negro*, or black micaceous schist, carrying 10 ounces silver and 3 pennyweights gold, is not worked.

The gold is saved on blankets in the mill at the mouth of the adit, and the mineral is concentrated on *mesa durmiente* by hand-shoveling, after which it is sent to La Linea.

Rio Sucio.—There are a number of important mines at Rio Sucio, in the province of Marmato, which was out of our itinerary. The chief owner of these mines is Sr. Chaves. His principal holdings are two gold-mines, one alluvial, the other quartz. Both were taken for debts, and were in an unprofitable condition, but after the expenditure of a moderate amount they proved a bonanza to their new owner.

Caramanta.—This district is reached in about 5 hours' ride from Marmato northward. A number of mining projects exist there, but our time permitted only a few moments' stop at two mills, right on the road-side. These, however, are typical of the equipment of the vast majority of Colombian mines.

The first of these is "La Union," owned by Medellin parties, who, in a granite formation, have run a 30-yard tunnel in on a small quartz-vein, worked by overhand stoping. The total labor-force consists of 4 men. They have 2 mills; the upper one, of 10 stamps, we did not see. A few moments after our arrival the manager, Sr. Angel Maria Ramirez, having replenished the mortars of the upper battery, which produces about 1 ounce per day, came down to look after the road-side mill, where 5 stamps were making 30 drops per minute, and crushing 30 wheelbarrow-loads, or about 1.5 tons, per 24 hours. On a double row of 4 blankets, each 3 by 16 feet, they save, in conjunction with the yield of a small arrastra, 5 *castellanos* (1 ounce Troy = 6.7553 *castellanos*) per day. In this mill the regular solid-steel stamp-head was varied by an ingenious arrangement whereby the shoes could be renewed as required.

A little further on was the mill of the Yarumal mine. The battery has 8 stamps, but for lack of water only 4 were running.

These were making 35 drops. On 3 blankets they save 5 or 6 *castellanos* per day of 18 hours, washing the blankets every half-hour.

The tailings are caught and piled for treatment in an arrastra which it is proposed to build.

The vein worked runs in width from 3 to 12 inches. The labor-force comprised 12 men. The mill-men could give us no idea of the length of the working or the amount of ore crushed per day.

Tolima.

The Department of Tolima lies east of southern Antioquia and a portion of the Cauca, on the eastern side of the central Cordillera. It contains many mines of silver, and of gold in gravel and quartz, for the development of a few of which English capital has been enlisted.

A number of hydraulic workings were affected by recent *débris* legislation for the preservation of the navigation of the Magdalena river.

The ancient Frias mine, which was abandoned early last century, was reopened about 1870, and is now one of the most important in Colombia. It ships to England for treatment silver-ore to the value of over $500,000 per year.

Mines in the Department of Antioquia.

For the localities described under this head, see the map, Fig. 1.

Manizales District.—The nature of the formation of this district is shown by its name, *mani* being the native term for granite. The city of Manizales (population about 20,000) is 7000 feet above sea-level, within sight of the snow-capped peak "El Ruis," in the chain dividing Antioquia from Tolima.

There are in this vicinity many gold-bearing veins, which, were it practicable to introduce proper machinery, would pay for working. A number of small mines are in operation, but the three principal ones of the district are the Diamante, Gallinazo, and Tolda Fria. These are interesting not so much on account of their actual production as in the light of their possibilities.

Diamante.—This mine, about 6 hours' ride from the city, is owned by a local corporation, of which the Jaramillo-Walker

brothers are the controlling factors. It comprises 6 claims, having a length on the vein of 3600 meters. The northeast portion crosses the range into Tolima. Our pocket-aneroid stopped registering at 8000 feet, but we calculated the elevation of the mine at something near 12,000 feet. It is very cold; the drippings from the flumes freeze every night; and we, who but a short time past had been exposed to the torrid sun of the Cauca valley, felt the change sharply.

The mine was opened up in 1884, and under a succession of managers made but indifferent returns. Don Nicolas Jaramillo, the present manager, took charge in 1895, after a flat failure in the installation of the Patio process by a former engineer had nearly ruined the company.

The lode bears N.E. and S.W., with a vertical dip, and varies from 3 to 4 feet in width. The gangue is a soft breccia of the granite country, and is impregnated with small veins of mineral, in quartz matrix, from 0.5 to 1.5 inches wide, carrying some free gold, gold in sulphurets, silver sulphides and a very little copper. All of the gangue carries some free gold; in places, the small veins bunch into one, a foot wide, which occasionally swells into lenticular masses to the full width of the lode; in which case it loses in proportionate value of contents. The walls in most parts are firm, the caps to sustain the lagging to catch the *débris* from the stopes simply being set in with 4-inch shoulders, and placed from 2 to 3 feet apart. There are 9 galleries on the lode, running in from convenient points on the hillside and then turning into the lode; the longest of these levels is about 300 yards. From these the ore is mined by overhand stoping. At irregular intervals *tambores* or air-shafts are opened up to the next gallery, 60 to 120 feet above, and excellent ventilation is maintained. About two-thirds of the mineral opened up has been taken out. The lode goes down with increasing richness in silver; at the surface, however, it shows free gold, with but a trace of silver. The ore is brought out in rude trams running on boards, with guide-stringers to keep them from running off the sides. The water in the galleries is ankle-deep or more, no drainage having been provided; the grade is very uneven, making it a hard task for the peons to push out a car.

At the dump the ore is hand-picked, the richer being sent on

ox-back to the amalgamating-plant, and the balance let down 200 feet on a wire rope to the stamp-mill.

The average product of the mine per day is about 7 tons to stamp-mill and 1 ton to "La Siberia," the amalgamating-plant. The total returns for 1896 were $35,880 (Am. gold); and figuring the working-days at 300, we find that $14.95 per ton was saved.

The water for milling is brought in a ditch 2.5 miles long. In the dry weather of our visit the supply was 1 cubic foot per second. The mill has an 8-stamp battery, moved by an overshot wheel 24 feet in diameter, the stamps making 49 drops per minute, falling from 2 to 6 inches according to the condition of the cams, which are of *guayaba* wood, set on a revolving cylinder in circles of 6. The steel stamp-heads weigh 120 pounds, and the wooden stems 280 pounds more. The die is of granite, packed with clay. The mortars are refilled whenever one of the central stamps, having a spike driven in it, lowers sufficiently to strike a signal-blade. The discharge is through a sheet-iron screen, with punctured holes of one-eighth-inch diameter.

No quicksilver is used; the gold is saved on coarse plush blankets, which the attendant is constantly washing in a vat at one side. Once a day the contents of this vat are washed in a small sluice, where the gold is caught in riffles, and the concentrated sulphides are panned out and sent to La Siberia. These concentrates amount to from 0.5 to 1 ton per week. Finer crushing and the use of amalgamated plates would save more free gold, and a vanner would greatly increase the quantity of the concentrates. At present the clean-up is from 6 to 10 *castellanos* per day. Including foreman, mill-man, miners and helpers, 14 men are on duty at mill and mine.

Up by the mine they are building a small brick furnace, after the model in use in La Bolivar mine, at Ibague, Tolima, which loads at the top of the long, sloping chimney, and partially calcines one charge while another is on the hearth. They expect to turn out 0.75-ton lots every 3 hours.

At La Siberia the ore is first dried in the drying-furnace, and then crushed in a dry battery of 8 stamps to pass a revolving 40-mesh screen. The roasting-furnace, of the same style as that

at La Amalia, is charged with 0.75 ton of powdered ore, which is calcined for 5 hours, driving off most of the sulphur and arsenic. Then 120 pounds of the following mixture is added: Salt, 25 pounds; powdered ore, 75 pounds; charcoal, 3 pounds. The whole is then roasted half an hour longer, when the charge is drawn and taken to the amalgamating-barrels. Of these there are four, each run by a 12-foot over-shot wheel at the end of its axle. A barrel-charge is 0.5 ton; 1 hundred-weight of chunks of iron is added to aid in mixing. After running 1 hour 150 pounds of mercury are added to the charge, with water enough to keep it a fluid paste, and the operation is completed in 24 hours. The resulting amalgam per barrel varies from 20 to 80 pounds; retorting leaves 19 per cent. of metal, which loses but 2 to 3 per cent. when melted into bars for shipment to Paris.

The force in La Siberia is 5 peons. With superintendent, 34 drivers, coal-burners, carpenters, smiths, etc., the full complement is about 50 men. Wages run from 20 to 45 cents (gold) per day and found. The higher figure is only made by laborers on task-work.

Sr. Jaramillo attributes not a little of his success to the order he maintains. Not a drop of liquor is allowed in the mine; no music, dancing or card-playing; and, if not on night duty, the men must be in bed at 9 o'clock P.M. at the latest. It strikes one as remarkable that, under such rules, he is able to obtain miners.

The product of this mine for 1896 was:

Silver,	$19,242
Gold from bars,	11,988
Free gold from mill,	4,650
Total,	$35,880

It is claimed that the vein can be traced a long way down the mountain-side, which descends at about 45°. The vein "La Union," of similar nature and assaying the same, which runs parallel near by, can also be followed down.

It would seem that, attacking these two veins at a point some thousand feet below, and building an adequate plant, there could be opened a mine the output of which would be large for a long time.

Gallinazo.—Having heard of this hydraulic mine, we expected to see work on a gravel-bank, and were somewhat surprised to find that the operation is slowly washing down a mountain of quartzite. The camp is located by the line of sluices just 8000 feet above sea-level. The climate is very agreeable and invigorating. The owners are a local company, and the mine is under the management of Don Sebastian Echeverri. With a force of about 15 men, who earn from 16 to 28 cents (gold) per day and found, he has in the past 4 years washed over 200,000 cubic yards, and taken out some $75,000.

There is a mountain spur, running down from the Ruis to S.S.W., which, for about a mile on each side from its foot, at the junction of the *quebradas* Chinehiuia and Fraile, is flanked by *molix*, or quartzite, which, in its richer portions, is honeycombed with rust from decomposed pyrites, and in others is only slightly stained. The latter carry not over 20 to 40 cents per cubic yard. This deposit varies in depth from 3 to 4 yards up to 20 yards. The *molix* merges into soft porphyritic bedrock, which, as it goes down, hardens and turns into granite. The face of the mine is about 20 yards deep, and carries from 50 cents to $1.00 per cubic yard. Tests on some parts of the property have shown as much as $5.00 per cubic yard.

The work of the monitor is assisted by blasting of the bank. The giant is a No. 2, using a 3-inch nozzle; 200 feet of piping bring the water under 100-foot pressure.

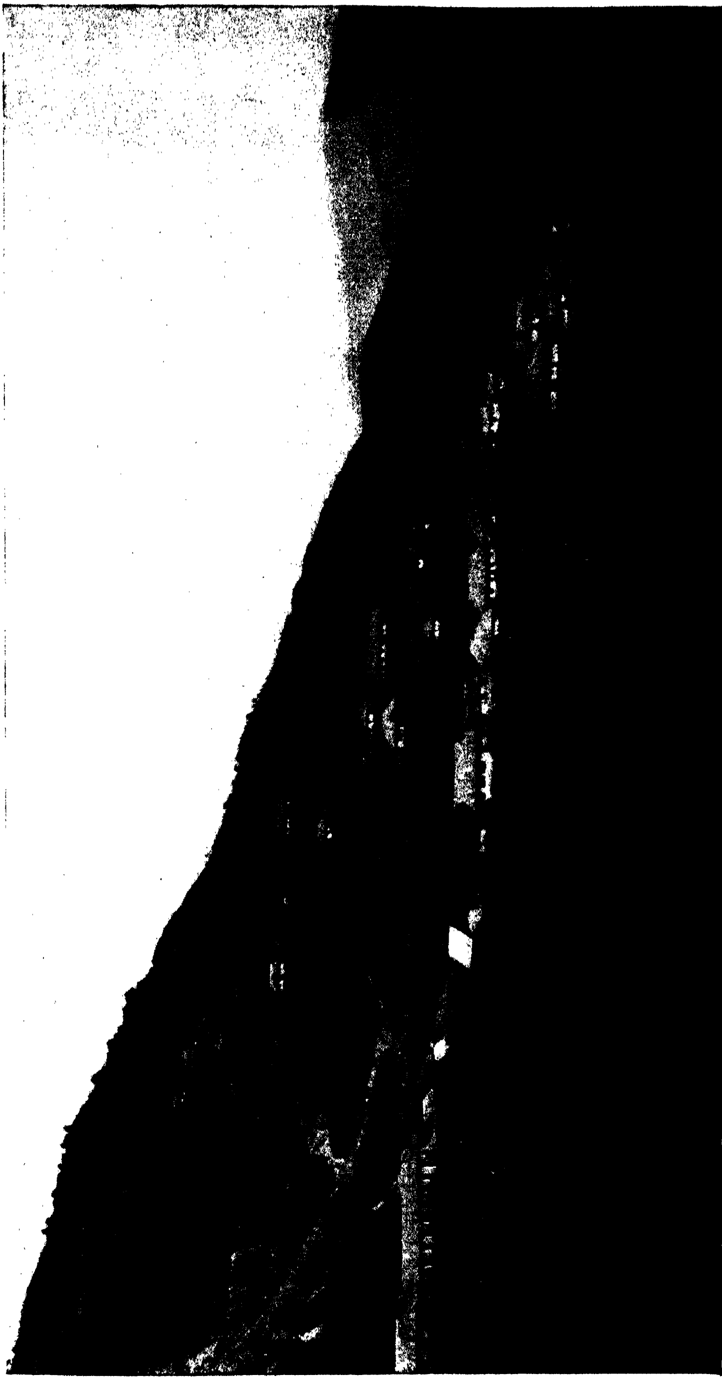
The penstock is at the end of a large flume-chute, and with more piping the pressure could be largely increased. The water is brought in a ditch some 3 miles long, and at small outlay could be doubled in quantity. In dry weather a smaller nozzle is used, and water from the Gallinazo creek is admitted at the head of the sluices, to carry the rock through.

The only gold saved is that liberated by the fracture of the rock and the breaking and abrasion in the series of drops to reach the sluices, and cannot amount to more than 40 per cent. of the contents of the ore.

The sluices are 200 yards long, 3 feet wide, paved with sawed blocks and set on the standard 4 per cent. grade.

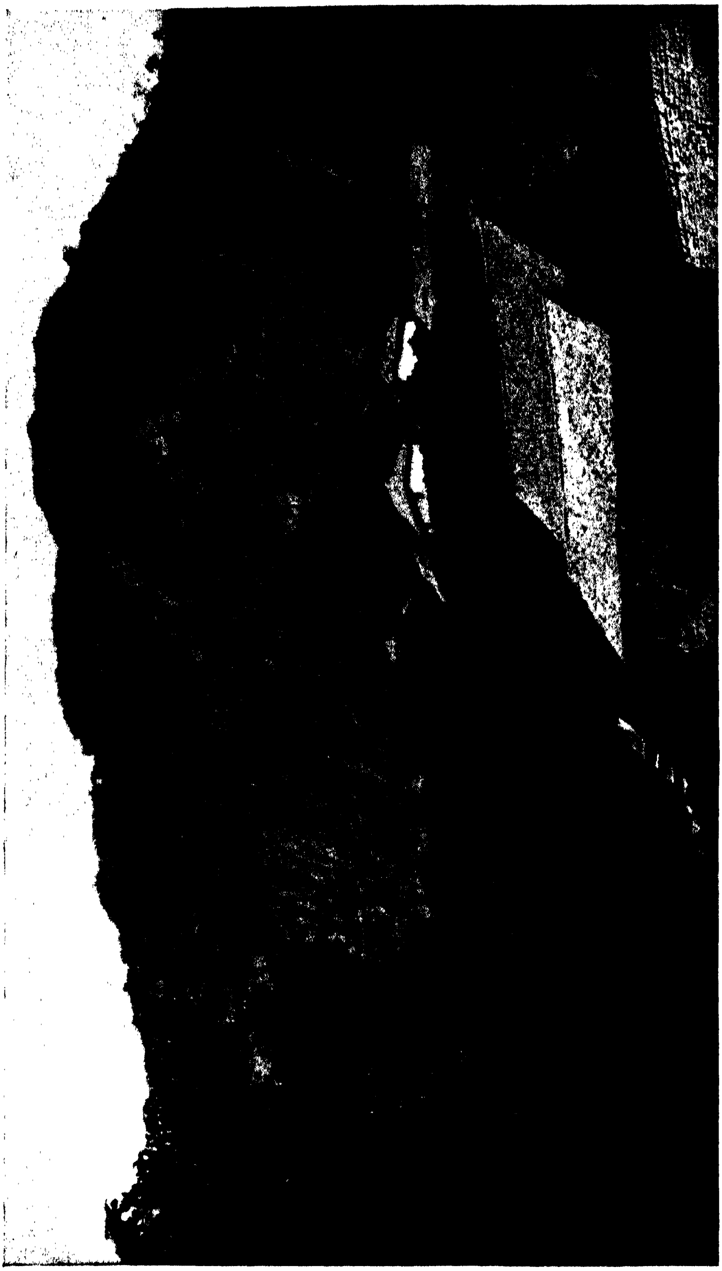
The gold is very fine, and doubtless a higher percentage would be saved by the use of the under-current.

FIG. 2.



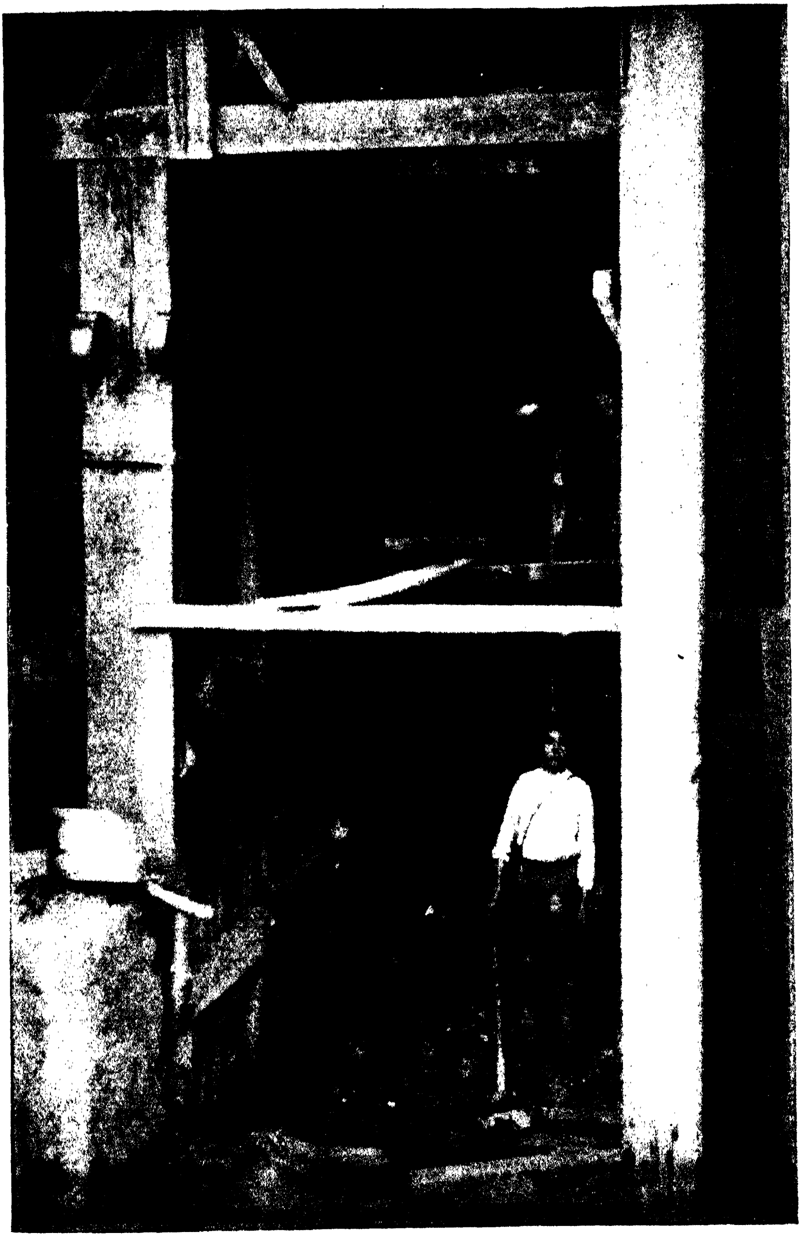
View of Marmato from the Plaza.

FIG. 3.



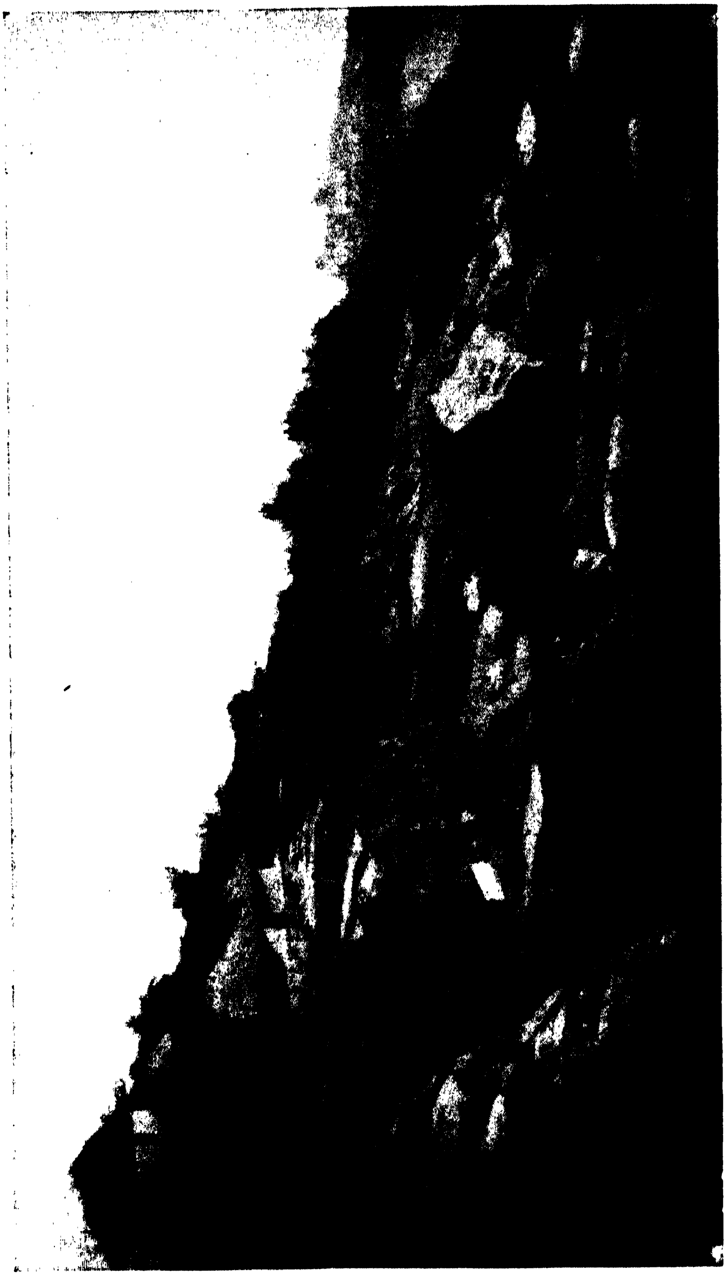
Cerro de Marmato; Mill in the Fore-ground.

FIG. 4.



Typical Native Colombian Gold-Mill (No. 3 at Marmato).

Fig. 5.



Gold-Gardens at Marmato.

The Gallinazo claims include about one-fifth of the ground showing this formation. Some of the remainder is not yet taken up, but the claims that are located are unworked because the Gallinazo Company controls the water.

Experience at Dahlonega, Ga., shows a profit in milling ores of even lower grade than these, when mined in this cheap way. It seems a pity that these properties are not combined, so that a plant could be located at a point where it could receive from sluices the whole deposit. Such a plant could be run by water from the Opallinazo creek; a pair of Gates crushers and some rapid-running Cornish rolls could reduce all the coarse material now going to waste, which, after passing over copper plates, could be concentrated on buddles to save the sulphurets for cyanide treatment.

Tolda Fria.—This mine is 10,000 feet above tide, in one of the most disagreeable climates imaginable. Every morning is bright and clear until 9 o'clock, when the clouds rise from the low valleys and envelope all in a cold, drizzly mist, which penetrates the marrow, and the effect of which is seen in the fact that every man and boy on the property is the victim of a continuous cold in the head.

Workings were started in 1873 and have continued ever since. The vein in exploitation is probably as small as any ever worked, having a thickness of but from 0.25 to 2 inches (average 0.75). The strike is N. E.—S. W. and the dip 23° S. E.

This *aguja*, or needle-vein, occurs in a talcose schist, the dip being parallel to the general lamination. The matrix is a highly crystalline quartz, showing free gold and rich sulphurets. Some pieces from the vein, which appeared to be masses of pure quartz crystals, on being ground proved rich in gold. The vein runs from $500 to $2000 (Am. gold) per ton. A large portion of this schistose formation carries from $1.50 to $2.00 per ton in free gold, being permeated with thread-like layers of quartz, which make up about one-fourth of its bulk.

The mining was done by open-work until 1891, when the amount of material which had to be removed in order to reach the vein became so large that underground work was started.

There are now two adit-levels running with the strike and numerous ramifications to take out the ore. These workings have a height of but 4.5 feet. They are carried by very stout timbering, filled in behind with waste matter.

The ore going to the mill is vein stuff (which is carefully taken out separately) and the rock enclosing it, which is more highly mineralized than the general mass.

The mine is unsurveyed, and no record is kept of the quantity of ore taken out.

Tram-cars running on 10-pound T-rails are used; but owing to bad grades and the condition of some parts, covered by slimy water, two men are required to push out a load of 9 cubic feet.

The underground force consists of 10 men, who receive 16 to 28 cents (gold) per day and found. From 50 to 75 pounds of high-grade dynamite are consumed monthly.

The two mills, one just above the other, are run by three men, who receive 28 cents per day each.

The batteries have 8 and 10 stamps, respectively, and the mills are of the general native type. The hard-wood stems last a year before splitting into uselessness; and the cams, which are set to give a 6-inch drop when they are new, are worn out at the end of 6 months, only being taken out when they no longer stir the stamp. At the end of each cam-drum (which is set in granite bearings and lubricated with water) is a wooden cog-gearing for running an arrastra in each mill. That in the upper mill has 2 stones weighing 2 hundredweight each, on a bed 4 feet 6 inches in diameter, while the lower one has a pair of stones weighing 3.5 hundredweight each on a 6-foot bed. These arrastras are used for grinding the concentrates from the blanket-washings, after they have been exposed to atmospheric action for 4 months. The plush blankets are set at an angle of 1° 45′, and are washed every 4 hours. They receive the discharge through plates punched with quarter-inch holes, which have splits allowing still coarser stuff to pass out with the slimes.

After passing the blankets the material reaches several boxes of sluices, 12 inches wide, with sawed checker-work for riffles. These are periodically cleaned, the gold is *batea'd* out, and the sulphurets are saved for the arrastra.

In dry weather the water-supply (1.5 cubic feet per second) only allows the running of 3 stamps in each mill. These make 39 drops, and can crush 600 pounds of ore in 24 hours per mill. After a rain the batteries run in full force at 68 drops,

and the two mills crush a total of 4 tons. For crushing the rich ore from the veins a "dolley" is used—a rude pestle made from a piece of bar-iron, flattened at the end and suspended from a vibrating pole. This works in a mortar slightly hollowed out in the earthen soil of the mill-floor.

Tolda Fria includes some 16 claims, and is owned by a Colombian company under the management of Sr. Marceliano Echeverri. All told, the force is about 50 men.

The total expenses are less than $600 (American gold) per month. From the time of its opening until last March the product has been over 2,345 pounds avoirdupois of gold .600 fine.

This mine is another instance of a formation that could be worked to advantage by a modification of the Dahlonega system.

Medellin.—This city of 40,000 inhabitants is the capital of the Department of Antioquia. There are no mines in its immediate vicinity, but it contains the headquarters and agencies of various mining enterprises of the adjoining districts.

The river Porce passes by Medellin. Its bars, together with those of the Nechi, into which it empties, furnish large quantities of gold every dry season. These are the only rivers in Colombia the beds of which, in certain sections, have been dried by the use of wing-dams.

As much as $30,000 has been expended in building dams and erecting Chinese pumps to keep the work dry, and the results have been always encouraging, and sometimes immensely profitable. The gold is occasionally found from the top down, but the great "pay" is in the streak lying next to the bed-rock. In one instance 200 square yards of ground yielded 10,300 ounces. Near Medellin, Messrs. Heiniger, Bachman and Arango are working a 4-foot vertical vein running N.N.E. to S.S.W., a contact-lode between granite and diorite, which carries free gold and sulphurets. They save $65 per ton.

Well-known Antoquian mining-districts are the Amalfi, Anori, Santa Rosa, San Pedro, Concepcion, San Carlos, Fredonia and Abejorral.

The Remedios district, in the northern part of the State, is attracting attention because of the shipments of the Frontino and Bolivia Mining Co., an English corporation, the operations

of which started in 1852, and were unsuccessful for a number of years, but were finally put upon a paying basis by Mr. Robert B. White. This company is said to have the only well-equipped gold-mill in Colombia. Messrs. Ospina Bros., through whose assay-office pass all the bars of the company, informed us that the present output, in bars, is at the rate of 22 arrobas (550 pounds avoirdupois) per month, not including the rich concentrates shipped to England for treatment, the results of which, they say, are sufficient to pay all operating-expenses. This gold is .550 to .650 fine.

Comparatively recent quartz-discoveries have drawn attention to the Western Cordillera, and led to the establishment of the mining *pueblos* of Andes, Urrao, Frontino and others.

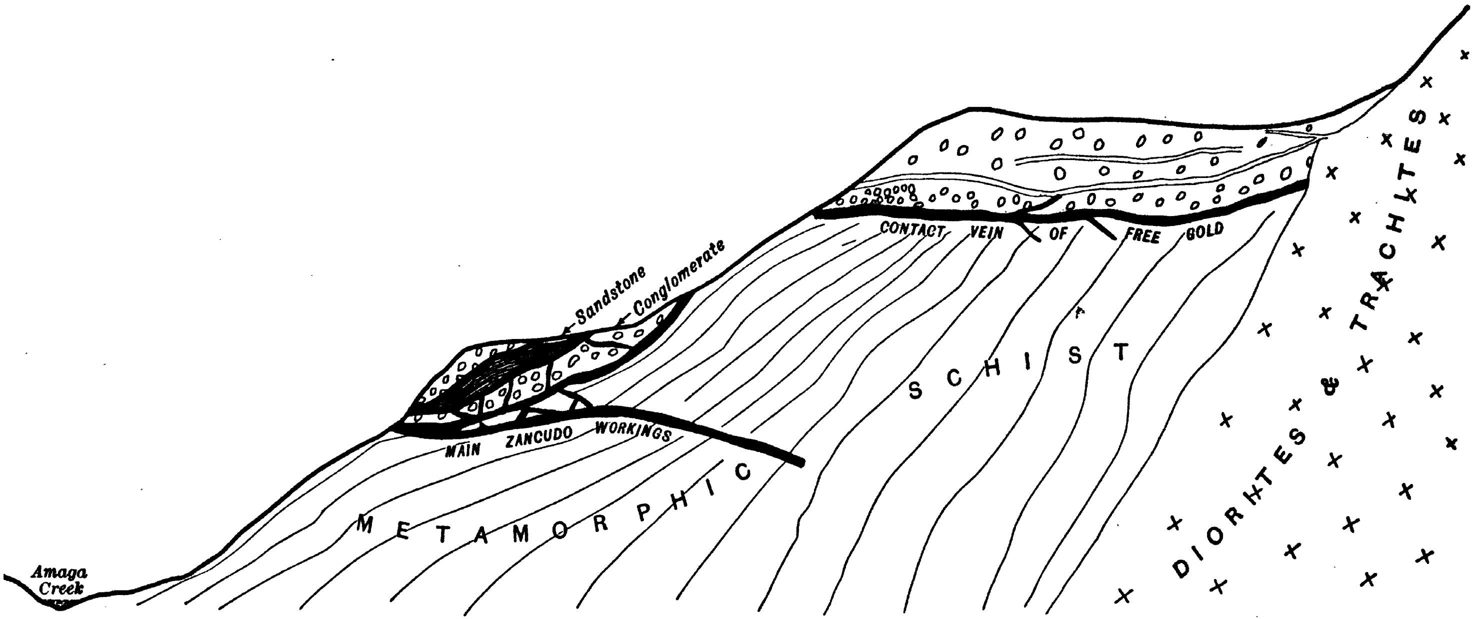
Titiribi District.—Zancudo Mine.—This mine, which was first opened in 1793, had had a checkered career under the management of native or European firms until 1883, when it was purchased by the present Zancudo Mining Co. of Medellin.

The property covers an area of 15 square miles, and includes over 300 claims, which are grouped under the names of Zancudo, Chorros, Cateador, Otramina, Balderrama and Encenilla, under the management of Don Carlos de la Cuesta. Don Severo M. Gonzalez is general superintendent, and has a large corps of assistants, the most active of whom at present is the consulting engineer, Sr. Juan C. Posada, who, after receiving his education in the United States, spent several years gaining practical experience in the mines and smelters of the West. Fig. 6 is a profile of the formation (not drawn to accurate scale).

The barometric reading at the head-camp, which is right over the main lode, is 4000 feet above tide.

The great body of the Cordillera is a fine porphyry or diorite, which breaks out from under an immense mass of metamorphic schist, supposed to be of Silurian age. The main Zancudo workings are on a contact-vein between this schist and a superimposed conglomerate formation. This vein runs N.–S., dipping 45° E., and the contact-formation runs out at a level 210 meters above the lower exposure. From the vein drop sundry off-shoots, which run together and form a main lode dipping 14° W. The workings have a N.-and-S. extension of about 1.5 miles; they are now in 600 yards, running W. on the main lode.

FIG. 6.



PROFILE OF THE TITIRIBI DISTRICT.

The contact-vein carries free gold and silver, as do the feeders to the main lode. After entering on the latter some 250 yards, the gold is found with the sulphurets. As the vein goes in, the silver holds, while the gold diminishes considerably in quantity. Associated with the gold and silver in these veins are arsenical pyrites, copper pyrites, gray copper, zinc-blende, galena, sulphurets of antimony, antimonial silver, and compounds of nickel, cobalt and manganese; the gangue is quartz and calcite.

Overlying the schist-formation, with a slight rise to the W., is a contact-vein under a similar blanket of conglomerate and sandstone, which is worked for free gold.

The vein in Zancudo widens at places to the full height of the drifts; in others it narrows to a few inches. The workings are surveyed every week, and orders for the stoping are given so as to maintain a reserve of 250,000 metric tons.

The average assay-value per ton is: gold, 26; silver, 550 grammes.

All ores are classified as smelting- and milling-ores. In 1896 there were extracted 4200 tons of the former and 32,000 tons of the latter. In the present product the average assay per ton of the smelting-ore gives: gold, 80; silver, 1600 grammes. The poorest ore which can be smelted at a profit averages: gold, 45; silver, 800 grammes per ton. The average assay of the milling-ore gives: gold, 15 (only 9 per cent. free-milling); silver, 300 grammes per ton. The poorest ore sent to the mill contains: gold, 6; silver, 180 grammes per ton. In the mill they concentrate from 12 to 15 per cent., losing from 3 to 6 per cent. The average value of concentrates is: gold, 55; silver, 780 grammes per ton.

The force employed by the company consists of 34 mill-men (paid 16 to 36 cents per day); 40 women (breaking and picking; sorters paid 20 cents per day, and breakers 10 cents per hundredweight); a force of 50 women for concentrating-work, receiving 16 to 20 cents per day; coal-miners and coke-burners number about 100 men, women and children, and earning 12 to 32 cents per day. These, with miners, foundry-men, carpenters, superintendents, etc., make a total of 1350, involving a daily expense of $480 (gold), while the average value of the daily product is $1000.

There is a foreman at each mine, under whom are the shift-bosses, miners, mill-men, etc. All work is in 12-hour shifts, and is maintained day and night. In mining, the contract-system is used almost exclusively, the company supplying tools and timber, while the contractor buys his dynamite and does his own timbering. In drifting they receive from $3.20 to $20 per linear yard, according to hardness of material. In stoping they are paid 60 cents to $2 per metric ton. Miners on wages earn from 28 to 48 cents per day.

Good grades are maintained in the galleries, and one man handles a loaded tram, being paid 12½ cents per load. Small boys earn 8 to 16 cents per day carrying the ore in barrows or packing it in hides from the stopes to the tram. The underground force numbers 300 to 350.

There are on the Zancudo two Ingersoll drills, operated by compressed air, only one of which is at work. It is used in the lowest level, called "La Troya," which is being driven to bottom all the stopes. The compressor is operated by a Pelton wheel under 150 feet head, with 300 feet of 11-inch piping and 1½-inch nozzle. Besides running the drill, it supplies air to 2 forges and to a large portion of the mine. The drill-man earns 56 cents per day, and his assistant 40 cents. The levels are carried 2.5 meters wide and 2 meters high, at a rate of 3 meters advance per month.

The milling-plant at Balderrama consists of a 6-stamp battery of the regular native type, after the old Cornish model. Chorros has 4 mills of 9 stamps each. Cateador has 2 mills of 6 stamps each, and, in addition, 1 iron mill of the California type, which is the first made in Colombia, having been built under the direction of Sr. Posada. It has 9 heads of stamps, in sets of 3, weighing 500 pounds each, with stem. The mortars, weighing 1500 pounds, are cast in one piece. The motive-power is supplied through a Pelton wheel. This mill makes per minute 85 drops of 7 inches, and crushes 2 tons per stamp per 24 hours, to pass through a coarse mesh. At present the ore is hand-broken and fed, but a Blake crusher is being built to add to this plant. Zancudo has 12 native mills, 6 of 9 stamps and 6 of 12, discharging through 16- to 20-mesh screens. All the stamps in the native mills drop from 8 inches to 1 foot, and make from 18 to 40 drops per minute, crushing

from 2 to 5 hundredweight per stamp per 24 hours. The blanket-system is used in all the mills, including the iron one. All tailings are run in troughs to one of the two concentrating-works, where there are double sets of 3 settling-vats each, the second receiving the overflow of the first and the third that of the second. The *cabezeras*, from the first vats, are coarser and richer in gold; the *medios*, from the second vats, are finer, and show more silver; while the *rabos*, from the third, produce slimes richer in silver. The upper works, which have been established 50 years, have 4 bumping-tables of the old German style for the sand and 1 for the slimes. A Frue vanner is being erected.

The lower establishment comprises two buildings, each having 4 bumping-tables and 5 *cernedores*, or concentrating-boxes, operated by means of shovels. Each table for sand is run at 18 blows per minute, and gives 0.75 ton concentrates per day. The table for slimes is run at 18 blows per minute, and gives 0.5 ton of concentrates per day. The tables are 9 feet long, 3½ feet wide, and 18 inches high in front, lowering to 7 inches at the rear. They are fed by a stream of water and tailings, which, after being taken from the settling-vats, are run through a mixer, where sufficient water is added to give the required fluidity. About 1 cubic foot per minute is the supply for each table. The tailings go from the tables to the women who operate the *cernedores*, where they are loaded at the head of the box and scraped up against the flow of water. An expert can concentrate a product of 800 pounds per day. The practice is to work in rows of 3; the first makes a partial concentrate, which is passed to the second, who betters it, and then to the third, who makes a clean product.

A *cernedor*, as used here, is 9 feet long by 18 inches wide and 18 inches deep, set on a 5 per cent. grade. The total product of concentration amounts to about 12 tons per day.

The two smelting-works are located respectively at Sitio Viejo and Saballetas. The latter will be abandoned upon the completion of improvements now under way at Sitio Viejo. We will, therefore, describe the former only.

Sitio Viejo has 7 roasters of 2 tons capacity each, 2 open-hearth or reverberatory smelters and 4 blast-furnaces, each of 10 tons daily capacity; also 3 cupellation-furnaces (German

type), with a capacity of 3 to 4 tons of lead in 3 to 5 days. The laboratory is located at this point. A new roaster of the long form is being built according to plans by E. D. Peters in *Modern Copper Smelting;* also a large open-hearth smelter after the Argo practice, described in the same book. The latter will have a capacity of 25 tons per day. Two duplex blowing-engines supply the draft for the 2 pairs of furnaces; one, the oldest, is horizontal, made of wood, and run by an over-shot wheel 17 feet in diameter. It uses 80 cubic feet of water per minute, and delivers 2000 cubic feet of air per minute at 8 ounces pressure per square inch. The other is vertical, has a Cornish balance-bob, and is run by a Pelton wheel 4 feet in diameter, which is operated with 80 cubic feet of water per minute under 50-foot head. This wheel also runs a 5-stamp dry-crushing iron-mill for crushing matte, the stamps of which weigh 600 pounds each and make 40 drops per minute.

The roasters are 8 by 8 feet, inside measurement. They load through working-doors and discharge through the back. The charge is calcined 8 to 12 hours, according to the proportion of sulphur. Concentrates showing 32 per cent. of sulphur are roasted down to 8 or 10 per cent. The fuel is 0.75 ton of coal per charge. This is the waste coal from the mines, and costs $1.20 per ton delivered. It is soft bituminous coal, clinking quickly, and is burned on a grate of 1-inch round iron bars. The matte from the open-hearth furnaces is also roasted here.

The open-hearth furnaces have a capacity of 4.5 to 5 tons per day. The material treated consists of roasted concentrates, raw pulverized ore, and slag from the blast-furnace as flux in variable quantity, the aim being to obtain a bi-silicate slag. From 5 to 7 tons of waste coal is used per day, and 5 charges of about 1 ton each are made. They are loaded from a hopper directly into the crucible. When the charge is fused and the slag completely separated, the matte is run out through the tap-hole, while the slag is skimmed out through the working-door. Sometimes 2 or 3 charges of matte are left to make a big tap. The matte obtained is crushed in the mill, ready for the blast-furnace.

The blast-furnaces have a capacity of 10 tons per day each, with a consumption of 2.5 tons of coal. They are rectangular, 5 by 2½ feet, inside measurement, with vertical walls 8 feet

high up to the charging-door, from which runs a 10-foot stack. The lining is a sandstone from one of the seams of the coal-measures. Air enters through 5 tuyeres, 3 at the rear and 1 on each side. The charge is made up as follows: Broken smelting-ore about 20, roasted matte about 14, unroasted blast-furnace matte about 35 per cent., and the rest slag from the same blast-furnace. The aim is to obtain a sesqui-silicate slag. The product of this furnace is treated with 30 pounds of pig-lead and litharge to each 100 pounds, and the bullion thus obtained is cupelled in the German furnaces. It contains 1.5 per cent. of gold and silver, the weight of gold being about 5 per cent. of that of silver. The lead used is imported at a cost of 8 cents per pound, delivered. The final product is melted into bars of about 25 pounds each. From 1883 to the time of our visit, last March, there had been melted 10,290 of these bars, of a total value of $2,670,000 (American gold).

The force at Sitio Viejo comprises about 80 men. Wages run from 24 to 48 cents, while smelters get $1.20 to $2.00 per day.

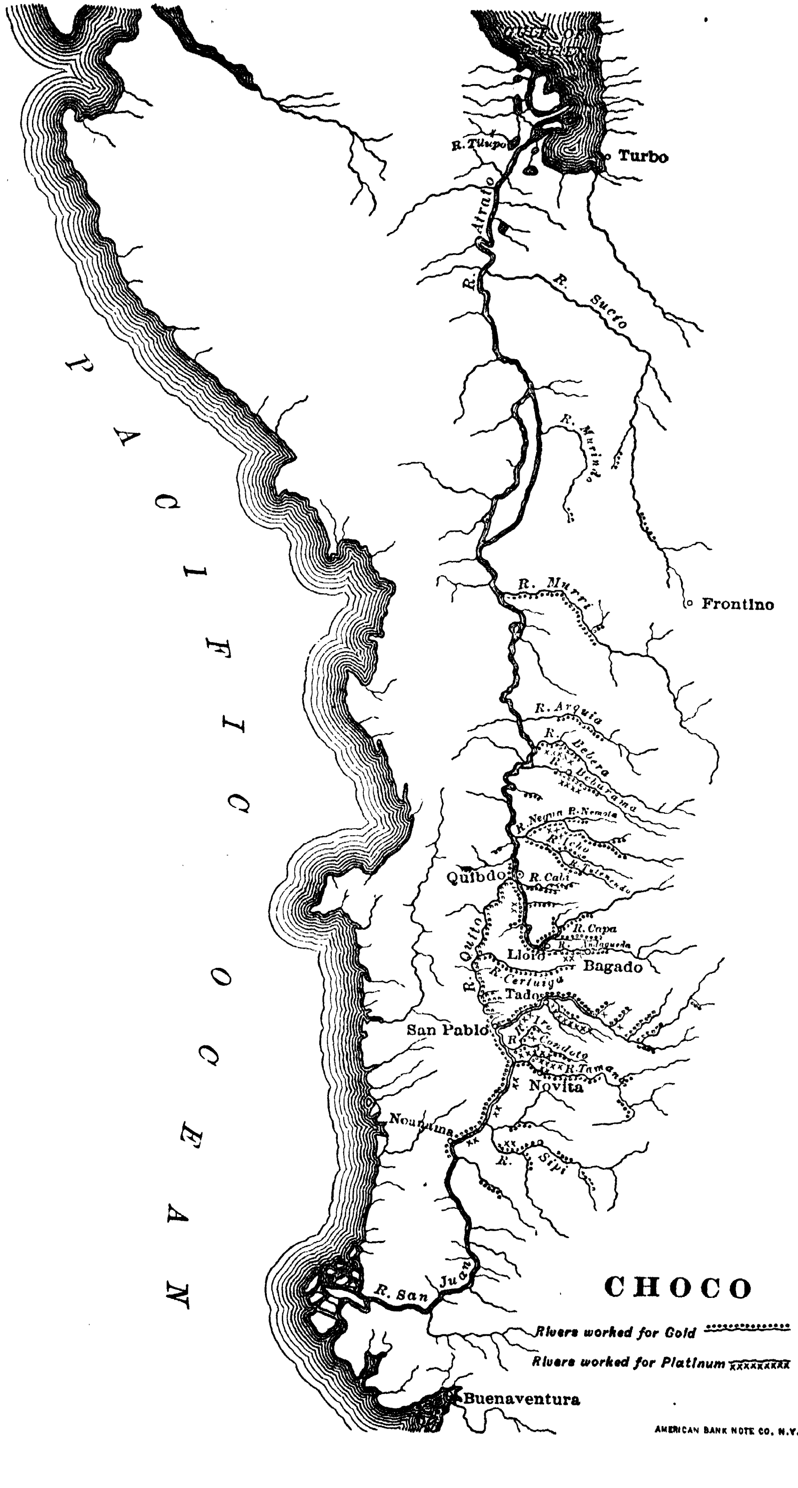
The coal used is extracted from a mine on the property, within sight of Sitio Viejo, across Chorros creek. It is delivered across the valley, a distance of about 900 yards, by a Hallidie wire-rope system, having a carrying capacity of 4 tons per hour. There are 18 seams, from 2 to 45 feet in thickness, aggregating over 100 feet, lying between sandstone and clay, with a 45° N.E. dip. The coal gives 48 per cent. of coke, which contains 10.8 per cent. of ash.

The company runs its own foundry and machine-shop, which supply all its own needs, and enable it to sell mills, stamps, Pelton wheels, Frue vanners, crushers, etc., to other mines. The pig-iron is brought from England, and costs, delivered, $120 (American gold) per ton. The company formerly made its own car-wheels; but as they lasted but a few months, against several years for the imported chilled ones, this practice was abandoned.

The Chocó.

Under this name is generally understood the territory drained by the rivers Atrato and San Juan and their tributaries. The climate of this region, by reason of the great humidity of the atmosphere and the decaying vegetation, engenders paludic

FIG. 7.



Turbo
R. Tilupo
R. Atrato
R. Sucio
R. Murindo
Frontino
R. Murri
R. Arquia
R. Bebera
R. Buburama
R. Nemota
R. Icho
R. Tutunendo
Quibdo
R. Cabi
R. Capa
R. Andagueda
Lloro
Bagado
R. Quito
R. Certugui
Tado
San Pablo
R. Iro
R. Condoto
R. Tamana
Novita
Noanama
R. Sipi
R. San Juan
Buenaventura
PACIFIC OCEAN
CHOCO
Rivers worked for Gold
Rivers worked for Platinum
AMERICAN BANK NOTE CO. N.Y.

fevers, which, however, are not deadly as a rule. These fevers can, to some extent, be avoided by care in diet and living; and foreigners who make periodic trips to a healthful climate, such as the Antioquian Cordillera, or the mouth of the Atrato, can live here in health.

Fig. 7 is a map of the Choco.

The first rich harvests of gold from this great district were from the beds of the creeks and small streams. Afterwards slaves were put at working the abundant alluvials. Since the abolishment of slavery the negro has remained practically master of the field, making his living by the *batea.* To this day gold-washing is the only industry in the Choco; and by the collection of a multitude of small individual contributions the gold exportation of this region is made up.

As the exports are made by a number of commercial houses, it is difficult to ascertain the present output, which, however, is estimated at between $300,000 and $400,000 per annum.

Many quartz-discoveries have been reported from the headwaters of the San Juan, and at one time some Antioquian miners crossed over from the eastern side of the western Cordillera and worked some outcrops by rude grinding on stones; but the heavy rains and the difficulty of obtaining supplies soon drove them from their work.

On our way from Antioquia we noticed several vein-formations crossing the trail which gave superficial evidence of deserving investigation. But we had no time to make any tests; the trail is in abominable condition, and had we failed to reach the rude shelter, with its surrounding small pasture, we should have had to pass the night in torrents of rain, and our mules would have died of hunger and exposure, as has frequently happened on this route. After five days of such travel, when the animals were constantly falling, and when it was necessary all along the line to send ahead a peon with a pick to cut steps in the steeper parts, it is not remarkable that the samples we hastily broke off with our belt-picks should fail to appear. These veins were in a granite country. Other indications were seen crossing a light blue porphyry. We also noticed outcrops in the metamorphic rocks about midway up the Andagueda; but these were not as promising in appearance as those seen on the mule-trail by the banks of the Atrato.

Antioquian rubber-hunters tell us that they see good-looking vein-outcrops near the head-waters of the Negua and Bebarra rivers. At Urrao, near the source of the Murri, quartz-ores are successfully milled, as also at Frontino, on the western slope of the northern end of this cordillera.

The lodes seen; the consideration of the milling development on the eastern or Antioquia and Cauca slope of the Cordillera; the massive formations of granite, porphyry, schists and metamorphic rocks, the continuity of which, throughout the whole slope, is proven by the gravels of the rivers which are drained; and, lastly, the placer-gold, which has evidently come from the denudation of quartz-veins in the Cordillera, are convincing evidence that the Choco is an extensive field for quartz-mining development.

The dense vegetation, the difficulty of transporting supplies, the exigencies of the climate requiring men of special endurance, all contribute to make such development a matter of slow realization.

A prospector in a cold climate can tramp all day carrying his kit, but climbing the foot-hills of the Choco produces a perspiration so profuse as to cause a faintness which prevents locomotion, and is only to be relieved by a hearty meal—a matter not always easily attained in a rainy jungle, where supplies must be securely packed in oil-skins, and are generally some hours in the rear. A white prospector must go with a train of negro packmen and others to cut trails, besides extra men to prevent his excursion being brought to a stop by some of his men falling sick. Thus it may be seen that quartz-prospecting in this region cannot be undertaken by poor men for many years to come. But groups of two or three men of due experience and solid constitution, equipped to stay in the wilds for six months, may go at the work with the assurance that they will discover many quartz-veins, and the likelihood that among them something will be found worth developing.

Many engineers, including some of repute, have visited the Choco, and all have subsequently published opinions very favorable to a great future prosperity for this district, backing their opinion by general statements based on isolated observations, which greatly exaggerate the real facts when thus applied to the whole region. Of such a nature is the statement

hat every yard of surface gravel carries an ounce of gold, or the report that the negro with his wooden pan makes $5 to $8 per day. Quite as inaccurate is the observation that panning on the river bars yields never less than $2 per day.

The great civil engineer Trautwine passed through the Choco about the middle of the century, making studies for an interoceanic canal. His attention was naturally drawn to the gold-washings; and he is quoted as having declared the belief that a railroad could be built through this section with the gold that could be taken from the cuts. We doubt the authenticity of this statement. It is difficult to conceive of a man of Trautwine's standing uttering such an absurdity. Moreover, we have seen for the alleged quotation no more reputable authority than that of the reckless promotors of a long-since defunct mining company owning a claim on the Andagueda river.

One of the most reasonable observations regarding the Choco was made by Mr. J. L. Hayward, who says that the results of some thousands of tests are a showing of over $1 per ton for surface-gravel of the river-bars. Our own tests show many places where gold is not found until from 3 to 10 feet from the surface, but there are so many more where fine gold is found from the surface down that Mr. Hayward's statement appears fairly reasonable.

Mr. Robert B. White spent some time testing the San Juan and some of its tributaries. He reported very enthusiastically, and were it not that his remarks are the results of actual tests, and but for his successful career as a mining engineer, it might be difficult to give full credence to his figures. He says:

> "I examined the sand of the San Juan and Tamana rivers at many points, and in many cases found it rich enough to pay for the labor of washing it by hand in a trough. . . . . From what I observed, and from what I know of the results of the work of the natives, I can say that the stratum or bed that rests on the rock produces, by careful washing, ten ounces of gold to the square yard. . . . . There is no doubt that at some points the yield may reach fifty ounces. . . . . I do not know any river in any country of the globe, outside of Colombia, where such favorable conditions for the extraction of gold exist."

The publication of highly-colored reports has brought strangers from time to time, who thought they were going to wallow in gold. Some 50 years ago a pretentious expedition arrived in a steam-launch, bringing a supply of scoop-shovels,

bags and bar-moulds. Remnants of the hull of their iron launch are still seen, but history has not recorded the number of bags of gold they carried away.

A little later a California miner named Stein arrived, and after spending some time prospecting located a high bar or section of the old channel of the Andagueda, where he built a pump, and by shafting and drifting took out some $50,000 in 4 years.

Attracted by Stein's success, some few others came and worked similar ground on various rivers with moderate degrees of success.

The white merchants of the Choco do not, as a rule, take to mining, except in rare instances, when business is dull. Under such circumstances Sr. Eladio Ferrer went to mining a claim on the Bebara river. After taking out a comfortable fortune he returned to the more congenial business of trade, having a branch house in Quibdo and locating himself at Cartagena. Relatives of his at Quibdo own a mine, "El Recuerdo," where in dull times Don Ricardo Ferrer retires with a few hands and puddles a fair income out of the clay.

The new-comers into the Choco, as in all new fields, are often, and, indeed, mostly, entirely ignorant and inexperienced in mining of any kind. They are apparently imbued with the idea that the inhabitants of the locality are not rich solely because they do not know that gold is yellow. The early Spaniards having cleaned out all the easy work in the rich creeks and gulches, and there being no established mines where they can gain experience while making a living, they are soon reduced to a sad plight.

The dry season, which permits panning on the river-bars, lasts only a short time, and there is no case on record where such adventurers have accepted the alternative of standing up to their necks in water, ballasted with a large stone on their hindquarters, and ducking under water, between breaths, to scratch up a panful of gravel, as is the practice of the negro miners.

The last party of this kind arrived in the Spring of 1896. It consisted of four young men who admitted their absolute inexperience, and another older citizen from a western agricultural State. The latter had represented himself to his companions as an experienced miner, while the trouble was that he

did not know the difference between a Frue vanner and a ground-sluice. After landing a fine outfit of tents, cots, picks, shovels, and a retort of 25 pounds' capacity, which they sent around in canoes, they took the cross-country trail to the village of Tutunendo, carrying little four-cornered bread-baking pans strung to their belts, in case they should see any particularly rich place where they could do a little panning, on their 5 hours' tramp. They pitched their tents on the banks of the Tutunendo, built 10 feet of sluices, and patiently labored 20 days at a piece of gravel on a high bed-rock that the negroes who abound on that stream had thought too poor to work. At the end of that time they made a clean-up, and not even finding the quicksilver with which they had loaded their box, decided to go home. This they did, selling everything but their shirts to pay their passage, and all of them except the expert, who died of a fever contracted on the way, doubtless reached home safely, and have since been made happy by "dollar wheat."

About 1887 a Boston company leased the mine "La Virgen Maria," on the Andagueda river below Bagado, owned by Don Leonte Castro and others. A hasty prospect showed very good results, and a pipe-line and monitor were put in. For a time all went well, and some 100 pounds of gold were said to have been taken out; but after working some time they found that the bed-rock faulted and rose in a high wall in front. Further work would have required more capital than the stockholders were willing to supply. On this property very little grade could be allowed for the sluices, owing to the flat nature of the ground; and in consequence they often choked up. It was their practice in such cases to simply turn the monitor down the line and drive the gravel through; an effective remedy for getting rid of, at least, the gravel. If, as the owners claim, a large area of this property is covered with pay-gravel, it would be a good place for Crandall's pumping and hydraulic elevator system, recently described in the *Transactions*.*

A Colombian company is running a monitor at the Chambaré mine, on the Andagueda; the bank is some 50 feet high, most of the gold being in the streak over the bed-rock, which

* *Trans.*, xxvi., 62.

is dipping in. This streak at first showed $1.50 to $3 per cubic yard. The whole amount of gravel worked, up to the time of their second clean-up, yielded an average of 19 cents per cubic yard. At this time the bed-rock was still dipping in, and the pay-streak showed rich. We washed 4 *bateas*, with an average of over $2.50 each.

It was proposed to work by drifting; but they had barely started when the bank caved, and as the dam had burst, there was no water to wash out the *débris*. Work is now proceeding slowly with wheelbarrows; but the owners are able or willing to advance but a few dollars per week, so the results are very meager. This property contains a considerable length of an old channel of the Andagueda, but no prospecting has been done to show its value. Not even a shaft has been sunk where they are working. The situation, but little above the present channel, and the in-dipping bed-rock, are favorable indications of a very good property.

In 1896 a party of Americans, Frenchmen and Colombians came from the coast to work in the bed of the Bebara river with a diving outfit. Only one of the men could go down. They soon got to quarreling among themselves, each alleging non-fulfillment of agreement against the others. Had they worked in harmony, the results of some $5 per cubic yard in gold and platinum which they had obtained before even reaching the bed-rock indicated that their enterprise might have met with a successful issue.

A French company, under the management of Mr. Ferdinand Mourgues, an engineer of large Spanish and South African experience, has been fitting up the Santo Domingo mine on the Bebara river. Mr. Mourgues had recently taken charge at the time of our visit. He showed us a large stretch of land which his prospecting showed to carry from 20 cents to $1 per cubic yard. It is comparatively fine gravel, with sandy clay as the greater part of the burden. The bank runs from 10 to 15 feet deep. The bed-rock, a soft sedimentary, allows of less than 3 per cent. grade to the sluices. A reservoir was being built to pipe water, under a pressure of 40 feet, to a small monitor, which the low height of the bank will permit to work effectively under this small pressure by keeping it close up to the face. This mine is now in operation, but the interesting point of a clean-up has not yet been reached.

The San Juan section is as rich or richer than the Atrato; but by reason of lack of steamer-service to reach it from Buenaventura, and the long canoe-journey from Quibdo to the post-trail across the isthmus which divides the waters flowing into the Atlantic and Pacific, it has attracted but little white enterprise since the days of slavery. In those days its alluvials were worked by large and important *haciendas*. Some ten years ago an American attempted to secure from the negro owners a lease of the gravels bordering on the Condoto river, but was unsuccessful in obtaining a reasonable working basis.

Latterly, Don José A. Salazar, a Cauca miner, has done considerable work in sinking wide shafts in a bed of low shallow gravel, taking out from a few ounces up to a couple of pounds of gold per shaft.

The negro mining may be said to be divided into the two general groups of land-mining and river-bed work. Taking the all-year total, more gold comes from the land-workings than from the gravels of the river-beds; but in times of drought men, women and children go to panning on the river-bars, and produce for the duration of the dry weather an amount much in excess of that of a corresponding period of work in the land-mines or in dry placers. According to local exigencies, these general groups are subdivided, the distinctive systems in use by the Choco miners being, in land-mines:

(*a*) Low flats, with bed-rock near or below the level of the stream on which they border. These are worked by large shafts from 9 to 18 feet deep, sunk in a rectangular form, a popular size being 10 by 20 feet. The pay lies in the gravel over the bed-rock, which will have a thickness of from 6 inches to 3 feet, being covered with barren soil. These shafts, when on very small flats bordering narrow streams, are located by long, wooden, iron-tipped sounding-rods or *chusas*. The operator distinguishes by the sound whether the soil overlies gravel or smooth bed-rock. Here, as in all their workings, the only tools are small crow-bars, one end pointed, the other flattened to a cutting edge some 5 inches wide; *almocafres*, or short-handled scraping-hooks; *cachos*, or pairs of wooden shells, used in throwing out *cascajo*, or gravel; and their *bateas*, in which they raise the gravel, bail, and wash the gold. Such workings

may be seen in the streams tributary to the Negua, Cabi, Condoto and Bebarama rivers.

(*b*) Flats where the gravel is unworked on account of being cemented, or for lack of grade, but where the overlying soil of 2 to 5 feet thickness is of a clayey nature and carries very fine gold. The soil is softened by throwing it into a ground-sluice of little current, then hand-puddled, and the entire contents of the sluice are washed by scraping down and panning. Instances of this nature are found on the borders of the Atrato below Quibdo and below Lloro.

(*c*) Large flats, such as on the Quito side of the Isthmus of San Pablo, where the streams are diverted to run along the foot of low banks, the soil and light sand are washed away, the stones thrown out, and the remaining heavy sand is panned out in the *bateas*.

(*d*) Gravel-flats of sometimes considerable extent, with bed-rock slightly above the level of the river on which they border. The banks are from 10 to 20 feet high, with little overlying soil. The streak over the bed-rock is the richest, and frequently the whole depth carries gold. Occasionally the gravel is deposited in well-defined layers, and contains three or four distinct pay-streaks. The topographical situation of such deposits is generally unfavorable to the bringing of running water, and they are worked by small reservoirs catching a limited supply of rain-water, which generally suffices but for a few hours' work, until replenished by the next night's rains. The water is allowed to trickle over the face, which the men cut down with bars, while the women and children pan out the gold in the ground-sluice. Such work may be seen along the San Juan, Condoto, Bebara, Upper Atrato, Lower Andagueda and Icho rivers.

(*e*) High bars along the upper course of the rivers, where the banks are higher and generally have a good dump and a live water-supply brought in ditches along the hills which back the gravels. Mines so situated nearly always handle much larger quantities of gravel than any of the others. The gold in them is practically all in the pay-streak and very coarse. We have noticed that frequently the low ranges backing these old channel-bars are covered nearly to their summits with gravel of older deposits, carrying workable quantities of gold.

Many of the land-mines or dry placers of the Choco make good showings, and it is very probable that careful prospecting would result in the finding of a number of good properties workable by either hydraulicking or steam-pumping, in connection with monitors, and, in many instances, hydraulic gravel-elevators.

While the Indians, Spaniards and negroes have cleaned up the easily-won wealth of the creeks, gulches and small streams, as well as large areas of the sand-mines, the rivers of the Choco are still the unscraped sluices of this extensive gold region.

During the short dry seasons the negroes work on the bars exposed on the sides of the broad channels of the Choco rivers in the following different ways:

1. In the Quito, the only river producing gold in any quantity not having its source on the slope of the western Cordillera, the gravel is quite fine, and the gold but little more than an impalpable powder. It is not known that any attempt has been made to get to bed-rock, the only method of washing being to scrape the surface of the *playas*, or bars, to a depth of from 6 inches to 1 foot and pan down to a concentrate the black sand, which is stored in a calabash and cleaned up at the end of the day. Being their own task-masters, the women (who are the chief miners of the Choco) very seldom wash more than 30 *bateas* per day, which yield them from 30 cents to $1.00 (gold). Gold-washing is done by groups of neighbors, whose necessities do not compel them to work either in the heat of the day or in rain, and many will not wash more than a dozen *bateas* in a day, spending a good part of the time smoking and gossiping. We have seen numbers of women, with nursing infants, waist-deep in the water, washing gravel—a remarkable evidence of the endurance of this race. The same bars are washed over and over, as every rise of the river causes agitation of the gravel, which brings a new supply of fine gold to the surface.

2. Bars which are worked from the surface down to bed-rock, the gold getting coarser as they go down, as is the case notably in the San Juan, Tamana, Sipi, Atrato and Murri, in which rivers the work is often done in the edge of the current, and where individual results of a *castellano* of gold per day are not infrequent.

3. Submerged bars (exposed parts not worked because of the excessive depth of the barren overlay), where they have to strip from 2 to 3 feet before reaching the pay gravel, such as in the Bebara, Condoto, Cabi, Icho, Andagueda, Tutunendo, Negua, Certuiga and other rivers; the gold found in this work is coarse, and the individual yield will average about the same as in the previous instances.

4. By far the most interesting Choco gold and platinum washing, both for the observer and for the negroes, is in places where the current has stripped a portion of the pay-gravel. The miners bind a heavy stone to the back of their waistbands and walk down the sloping bank into a depth of water of from 9 to 15 feet, load their *bateas* with as much gravel as their power of suspending respiration will allow them to scrape up, and, after coming to the surface two or three times for a fresh breath, finally bring up a full *batea*, which is washed by another while the diver takes a rest. This process is very exhausting and a dozen *bateas* is a good day's work. Such *zambullideros* or diving-places are worked on the Condoto, San Juan, Tamana, Sipi, Capa, Andagueda and Atrato rivers, and seldom yield less than a *castellano* per day, while an ounce to the individual is by no means a rare result. In times of exceptionally low water, such as only occur once in the course of a number of years, when the diver is able to go nearer the center of the channels, individual results of as much as a pound a day have been gained in numerous instances.

From 1882 to 1885 various attempts were made to dredge the Atrato. First a small dredge armed with a centrifugal pump was brought out; but defective construction ended its career within 6 hours from starting. Next a large dredge was put together at the mouth of the Atrato and started for the gold-regions; the peculiarities of the Choco climate make it difficult to secure timber which will remain sound, and the hull of this dredge was so rapidly rotted and worm-eaten that it sank and was lost when about half-way to Quibdo. Then a large dredge, called the "Cauca No. 2," was equipped with a vacuum-pump. This had an adventurous life of but a few months. We are told that no experienced miner was aboard the dredge; there were no spuds or any other appliances to maintain the hull at a level, and this had a disastrous effect on the operation

of the sluices, which were set on the hull. The pump worked very irregularly, sometimes bringing up the gravel so rapidly as to weigh down the bow to the water-level; the sluices were set to operate on this level, and when the suction-pipe struck compact gravel it worked very slowly, the bow rose, and gold, gravel and mercury were washed out of the boxes. Then a patent amalgamating-plant was brought out, a beautiful combination of sizing-drums, an iron tub with copper plates and jets of water. Then the vacuum-pump burst, and a brass-lined centrifugal pump was put in place. The suction-pipe was continually getting choked with stones that had to be broken out, but for eight days the results were said to be quite satisfactory; but by that time the lining of the pump had become worn to such an extent that it brought up nothing but light sand and water.

The total gold saved during these intermittent spells of operation amounted to but 16 pounds; and, no more capital being available, the project was abandoned. One of the mechanics employed on this dredge told us that when the suction was from gravel a couple of yards below the surface (comparatively barren stuff), average pans from the tailings that were returned to the river showed 10 cents in gold. If this be the case, comments are unnecessary regarding the effectiveness of the patent amalgamation.

Pumping-dredges have their advocates for light sandy material, where the gold is fine, but they are not qualified for work in the compact gravels of the river-beds of the Choco. The same may be said regarding the chain-bucket type used with such success in the light gravels of New Zealand and Montana.

What is wanted for the Choco is a strong, simple dipper-dredge.

The advantage of steady discharge claimed for the pumping- and chain-bucket systems can be secured for the dipper-system by having the sluices on a separate float, and the dipper emptying into a hopper at the head of the line. By means of an auxiliary stream playing into the hopper, the regularity of the feed can be perfectly governed. By having the head of the sluices at sufficient height to allow drop for a couple of grizzlies the heavier part of the cargo can be thrown out, and a lower grade allowed for the balance of the line, while the gravel from

the grizzlies could by a very moderate flow of water be carried astern in narrow, smooth-lined boxes.

The success of a dredging-plant of this system, built by the Marion Co. and owned and operated by the Messrs. Birch, and a similar Bucyrus plant, owned and operated by Mr. H. D. Jaquish, on the Chestatee river, of Georgia, has aroused a new interest in the Choco, and most of the river-bed claims of this district have been recently taken up by various Colombian, American and English parties for working by this method.

Mining Laws.

The mining laws of Colombia are very equitable and well calculated to promote the establishment and growth of mining enterprises, besides giving full protection to the capital invested.

The size of a claim is, for a quartz-vein, 600 meters long by 240 meters wide. To the first discoverer in a new district, one payment of the tax is credited for three claims. In alluvial mines, claims are of 5 by 2 kilometers. Claims may be located on cultivated lands by giving security for any damages that may be caused. For both quartz and alluvial mines there is a reasonable initial fee and a small annual tax. To secure a claim, the mode of procedure is to first give notice to the *alcalde* (mayor) of the district, specifying the exact location. This is recorded and a copy is furnished, which is sent, with a memorial, to the Governor of the Department. In due course papers arrive at the local alcalde's office, decreeing possession. These are posted for three weeks. If there is opposition, the case has to be tried in court before proceeding further. If no one opposes (as is usually the case, when care has been taken in locating the claim), possession is then given and the papers returned to the Capital, whence, in course of time, are sent the titles. Owing to lack of speed of communications, it takes some months to receive title when the claim is at a distance from the Capital. By a recent law passed in the interest of mining companies owning a number of claims but only working one at a time, parties paying their taxes retain their title, even if they do not work. In view of the delays in receiving supplies and the difficulties of travel, this is a very attractive feature to the foreigner who proposes locating in Colombia.

The Government and the People.

The foreigner domiciled in Colombia enjoys the same rights, save as to suffrage, as the native or naturalized citizen, and receives the fullest protection of life and property. Honesty is the rule among the poorer classes, while the merchants are noted for their high standard of honor in business transactions.

As a result of the high-grade educational institutions of the country, and of university training and travel in the United States and Europe, many Colombians have developed a high degree of culture and scientific attainment, and the foreigner properly credentialed receives from them cordial treatment.

The Future.

A number of opinions have been expressed in the past to the effect that Colombia gold-mining will develop "booms" surpassing in magnitude the "rush" days of California and Australia.

In a magazine publication of recent issue, a well-known railroad and United States Government engineer states his conviction that Colombia is entering on a "boom" which will rival those that have been seen in Australia and the Transvaal, and cause the world to behold the results with unbounded amazement. This unwarrantable talk frequently comes from people who have never even visited the Colombian gold-fields, and in many instances from men who, if they have seen a portion of the auriferous sections, are not qualified to express a valuable opinion. Such statements only serve to deceive poor dupes who come out with their heads full of extravagant hopes, to be inevitably stranded.

The publication referred to gives some remarkable test-results, some or all of which may be true, but which need an explanation of the circumstances in order to merit the serious attention of men who are not credulous enough to believe that a native population of able miners will work for 30 cents per day when gravel carrying $200 per cubic yard lies at their feet.

Such a period of excitement can never come to Colombia, because the day is two centuries past when the poor man could sluice, cradle or pan out his ounce per day. It is the rush of such men under such conditions that starts all "booms."

Colombia is not now a country to be discovered. The amount annually produced by its mines has made the country known for hundreds of years.

Future developments will be brought about by the explorations of prospectors who can stand a monthly expense of, say, $300 (gold) for a matter of one, two or three years. Experienced men of sufficient means and physical powers will not come in hordes, but they will come; and their patient, hard work in the recesses of the Cordilleras will promote the growth of mining in Colombia.

Another source of increase in the Colombian output lies in the thorough examination of known mines, hitherto unworked, and the careful study of modern processes in successful operation elsewhere, applicable to such mines, and then their installation and operation by men experienced in the treatment of the peculiar conditions of each mine by the chosen method.

Another factor in the development of the mineral industry in this country is the examination of mines now in operation, and, upon assurance gained of their continued production beyond the shadow of a doubt, their purchase and the enlargement of their usefulness by the spending of sufficient money to equip them to best advantage, and put them in charge of sober, capable managers. Colombia is not a country in which to try experiments in either methods or men. So much time and expense is required to freight and place a plant in position that, no matter what guarantees might be given of a new process by the manufacturers, none but well-proved machinery should be brought here. It is a noted fact that whenever a man who has yet to learn the a b c of mining is sent to take charge of a mine, as unfortunately sometimes happens, he brings with him the prospectus of some new machine which is to revolutionize washing, mining or milling, and with which he is desirous of equipping his enterprise.

Colombia, under her wise laws and fostering policy, presents an attractive field to the American engineer, and energetic exploration, cautious investigation, the judicious expenditure of capital and good management will result in a healthy, substantial growth of her mining industry that will greatly augment her already respectable annual quota to the world's gold-supply.

Kalgoorlie, Western Australia, and its Surroundings.

BY GEORGE J. BANCROFT, DENVER, COLO.

(Atlantic City Meeting, February, 1898.)

WESTERN AUSTRALIA (often popularly called Westralia) comprises all of the Australian continent west of the 129th meridian. The latest census, that of 1895, gives it a population of 101,235 persons. It includes within its boundaries 975,920 square miles. About half the colony lies within the tropics. The climate varies but slightly, considering the vastness of the area. The interior is dry and hot, and subject to very sudden changes. The coasts are warm, and the rainfall abundant, varying from 20 to 40 inches per annum.

Perth is the capital and principal town, having a population of 20,000. Fremantle (pop. 9000), distant 12 miles, is the port of Perth, but because it is out of the track of the mail steamers, the mail and passengers are landed at Albany (pop. 3000) on St. George's Sound. A railroad, 338 miles long, connects Perth with Albany, and the running time of trains is about fifteen hours.

Kalgoorlie is only about 200 miles from the coast at the little village of Esperance; but to reach Kalgoorlie one must land at Albany, travel to Northam, within 66 miles of Perth, and then east at an acute angle 325 miles to Kalgoorlie; in all about 600 miles.

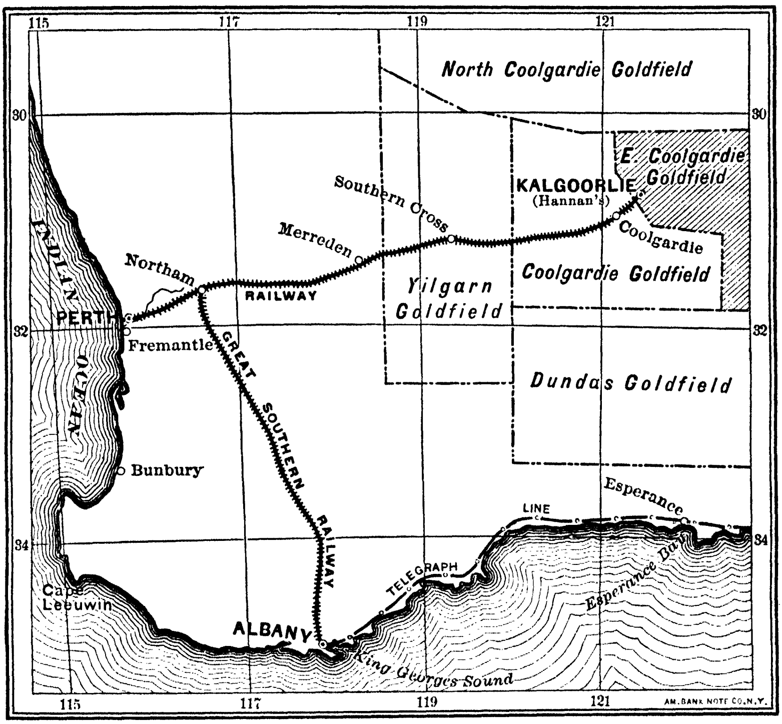
From Perth a railroad also runs to the Murchison gold-fields by way of Mingenew, Mullewa and Yalgoo, to Cue. A branch goes down to the port of Geraldton. The government operates all the railroads and telegraph lines, and has been very energetic, all things considered, in extending them into the gold-fields. As to the accommodations afforded by the railroads, the less said the better. The comforts of travel are conspicuous by their absence.

Of the 975,920 square miles of Westralia, the government, in its magnificent way, declares 256,060 to be gold-bearing.

There are the Kimberley fields in the north, of 47,000 square miles; the Pilbarra on the northwest coast, of 45,600 square

miles; the Ashburton, in the west interior, of 8200 square miles; and the southwest gold-fields, of 155,260 square miles, which include the Yalgoo, Murchisons, Coolgardies, Yilgarn and Dundas fields. It is with these southwest fields, and particularly the East Coolgardie field, which contains the famous camp of Kalgoorlie, that this paper deals. The position of the district is shown in Fig. 1.

FIG. 1.



LOCALITY-MAP OF THE
SOUTH WESTERN PORTION OF WESTERN AUSTRALIA.
SCALE OF MILES
0 50 100 200 300

The southwest gold-fields are a vast plateau of granite and paleozoic greenstones, flat as a table and dry as a bone. Here and there domes of granite, polished by the wind-driven sand, or black ridges of "iron-stone," rise out of the flat landscape. In places there are extensive dry-lake beds with scattered pools of very salt water, while still more rarely are found "clay-pans" and granite "water-holes," containing comparatively fresh water. It is a fact worth noting that the rock-enclosed

water-holes are entirely in the granite. There are no running streams at all in the interior. Those marked on the map are simply ancient river-channels, which have not become entirely obliterated.

Few areas of the same size as the Westralia gold-fields present such a monotonous continuity of the same geological conditions. Granite and greenstone are the prevailing rocks everywhere. As a rule, they run in belts trending north and south. There are a few lacustrine deposits of sedimentary rock; and towards the southwest the plateau, and with it the interior gold-region, is terminated by a depression, in which the principal rocks, according to Mr. Victor Streich, are of Mesozoic age.

Among the more recent agencies which have produced the perfect flatness of this plateau, the high winds have undoubtedly played an important part; and so marked is their work, that in many places the ancient drainage-channels have been obscured.

The fields are from 1000 to 1200 feet above sea-level. The climate is dry and hot, and would not be unpleasant but for the terrific dust-storms that blow for days at a time. The flies are a pest. A common complaint is "bung-eye," which is a swelling of the eye-lids, resulting from the unrelenting attentions of the bush-fly.

The rain-fall was 2⅜ inches in 1894, which was a very dry year; the average is about 4 inches per annum.

While the gold-fields are practically a desert as regards rain-fall and climate, they present anything but a desert appearance; for nearly the whole plateau is clothed in the beautiful verdure of the mulga (acacia) and salmon gum-tree (eucalyptus salmonophloia) which remain green the year round. At the end of the rainy season the ground is carpeted with millions of pretty "everlastings." The perfect adaptation of the flora to the severe conditions existing, suggests to me that the climate must have remained as it is now for a very long period of time.

The first practical discovery of gold in Westralia was made about 1882, by Mr. Hardman, in the Kimberley fields. In 1887 gold was discovered near Southern Cross, and from this discovery the gold-mining industry of Westralia dates.

The total gold-production of Westralia for 1896 was 281,263

ounces, and that of 1897 was 678,426 ounces. The output for the month of September, 1897, was 59,584 tons (of 2000 pounds) of gold-bearing rock, which yielded 78,281 ounces gold, giving an average of 1 ounce, 6 pennyweights, 7 grains per ton. Of the above amount Kalgoorlie produced 16,683½ tons, which yielded 33,848 ounces, or a trifle over 2 ounces to the ton.

In Australia they have a most unfortunate way of saying *gold* when they mean crude bullion, and such a magnificent disregard for trifles that they never mention the fineness of the so-called gold. Consequently, the above figures are absolutely worthless as they stand. However, the bullion is nearly all of high grade, and the average of 200 assays, which I have at my command, is $18.42 per ounce. Calculated on this basis, in dollars and cents, Kalgoorlie's output for September is $623,-480.16, and of this amount all but $3,131.40 or $620,348.76, came from a diamond-shaped space, about one square mile in area.

Kalgoorlie.

The first discovery of gold was made on June 14, 1893, by three prospectors, named Patrick Hannan, Dan Shea and Flannigan. Hannan lost no time in going to Coolgardie and lodging his application for a "reward-claim" (*i.e.*, a free-hold title to a claim). Hence the camp is frequently spoken of as Hannan's.

As usual in most gold-camps, the alluvial gold was first discovered. The ground was worked by dry blowing, and was found to be very rich. Figs. 4 and 5 illustrate the method of dry blowing.

In connection with the alluvial gold, I wish to call attention to one or two interesting points.

The veins contain gold either as a telluride or as fine "paint-gold" (very minutely divided gold), resulting from the decomposition of tellurides, but the alluvial diggings contained principally *coarse* gold, several nuggets weighing as much as 24 ounces having been found. Again, the richest alluvial diggings were on the north end of the field, extending clear to the top of the highest hill in the vicinity, while the richest lode-mines are found at the southern end of the field, on lower ground. There is evidence to indicate that the gold has not been moved any very great distance.

These facts suggest that the solutions rising through the fissures at the north end of the field failed to find a precipitating-agent until at or near the surface, and that there the conditions were such as to precipitate the gold in nuggets—conditions which very evidently did not obtain lower down in the veins.

I wish that I could furnish complete and accurate information as to the geology of this most interesting gold-camp. Unfortunately the government geological survey has only just begun its work, and the engineers of the camp have been so occupied with the commercial questions at issue, that they have had little time to investigate the rather obscure geological features. I will, however, briefly describe what I have learned from the best-informed men in the camp, and from my own observations.

Mr. S. Göczel, government geologist, in his report for 1894, on the Coolgardie district, thus describes the camp:

"A large break in the country, consisting principally of coarse-grained diorite, extends over 6 miles from N.N.E. to S.S.W. near the township of Kalgoorlie, and encloses a countless number of lodes. The line of that break is marked by low ironstone hills."

At first sight there does not seem to have been any local eruption to which we might attribute the conditions which have mineralized the veins. The diorite extends for miles in all directions. But there are two dikes, one at the southwest corner, and one at the northeast corner of the field. They are at some distance from the rich mines, but pieces of float which I picked up indicate that there may be others nearer the center of the camp. Prof. J. F. Kemp of the Columbia School of Mines has determined the dike-rock to be a pyroxenite, composed mostly of tremolite or actinolite.

Associated with the veins are lenticular bodies of graphitic slate. These have been called eruptives by some; but Prof. Kemp has determined the rock to be a true slate, probably an altered and metamorphosed clay-rock.

The rich area of Kalgoorlie is characterized by a bluish tinge of the rocks, while the rocks of the barren ground surrounding it have a greenish tinge. The local mineralogists have considered the two shades as different alterations of the same rock, and have given the name diorite to both.

It has been suggested that the blue stone is a later eruptive,

and that to the influence of this eruption the mineralization of the camp was due. Prof. Kemp has determined the blue stone to be a much-altered basic eruptive, which is now serpentine and chlorite. The green stone, as determined at Perth, is essentially the same, except that there is a little olivine present. Both rocks are so greatly altered that it is difficult to determine exactly what they were originally, so that there is no conclusive evidence that the blue stone is, or is not, a later eruptive.

The veins are mineralized bands in a schistose country. One of the features of the camp is the overlapping of long lenticular formations of all kinds. Fig. 2 represents patches of slate in the Lake View South mine. The same thing is seen again

FIG. 2.

LENTICLES OF SLATE, LAKE VIEW SOUTH MINE.
100 FOOT LEVEL; No. 3 SHAFT, SOUTH 50 FEET.

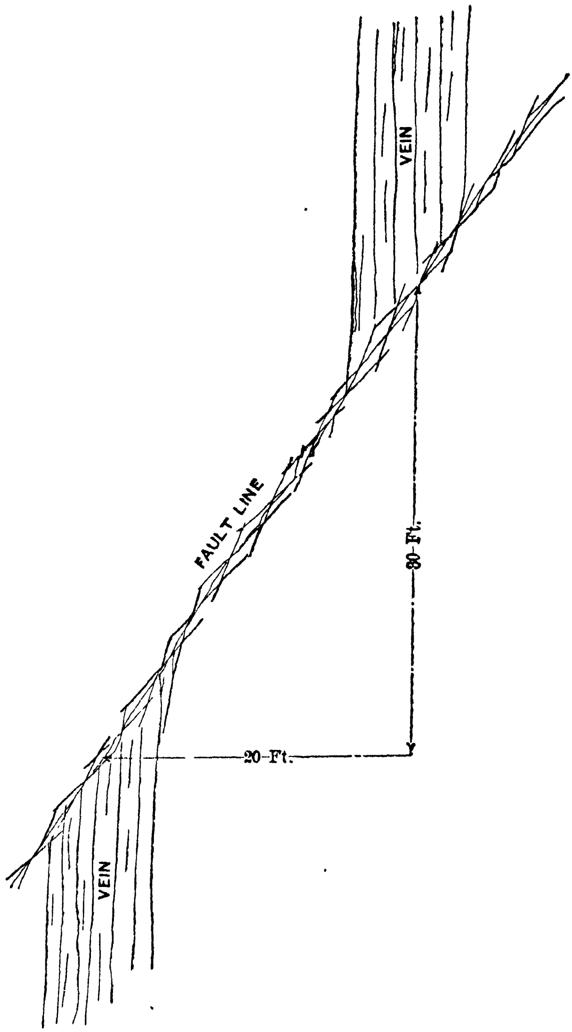
and again in the ore-bodies, it being a common rule among the miners to cross-cut east in going south and west in going north, when the vein pinches. The miners say the vein is "thrown" or "slipped," but there is no fault evident, and I consider it simply a case of co-ordinate fracturing which characterizes the camp. The same thing occurs frequently throughout Westralia, and I have noticed it in Rocky Bar, Idaho, and in Bear Creek cañon, Colorado. Often the rock between the pinched ends of the ore-bodies is stoped out with the rest, and the result is that there is simply a slight turn in the drift which obscures the true formation to the casual observer. Another feature is the reverse faulting shown in Fig. 3.

The ore is very slightly-altered country-rock containing iron pyrites and tellurides of gold. Calaverite has been isolated and proved by Tinley and Holroyd and others. It is probable that sylvanite, petzite and native tellurium also occur. Mr. H. C. Hoover reports the existence of sulpharsenide of gold in the Brown Hill mine. One of the interesting phenomena of the ore

is the occurrence, once in a while, of small particles of crystalline gold, surrounded by crystals of tellurides low in gold.

Owing to the fracturing of the country, there are "walls and

FIG. 3.



REVERSE FAULT. GREAT BOULDER MINE
200 FOOT LEVEL HORSE-SHOE VEIN; MAIN SHAFT; NORTH 320 FEET

walls," and often the pay-ore stops at a defined wall, and often it does not. Sometimes an ore-body is found by cross-cutting which did not appear at a higher level, and sometimes an ore-body disappears with depth.

Fig. 4.

Dry Blowing, with Batea.

Fig. 5

Dry Blowing, with Apparatus.

FIG. 6.

WATER-CONDENSING APPARATUS, WESTERN AUSTRALIA.

The ore-chutes of Kalgoorlie are remarkably long and continuous, and of high grade. The strike of the main veins is 40° W. of N., and, taken as a whole, there is no marked prevailing dip. Besides the main series, there are a few veins trending about 50° W. of N., instances of which may be seen in the Great Boulder, Boulder Main Reef and Lake View South.

I noticed two instances in which a vein of this class cut directly across one of the main series, showing the main vein to be of earlier origin. No change in the value of either vein was caused by the intersection.

Then there are quartz-veins running E. and W., which are generally barren; also narrow, flat quartz-veins, which are often very rich; and quartz-stringers, likewise rich occasionally, which run in apparently any direction.

In the Oroya and neighboring mines, in the northeast corner of the rich area, the veins dip sharply to the west at an angle of 45°, and the strike is about 20° west of north.

The ore is soft, but tight, and makes mining very expensive under present conditions. Drifting costs from $11 to $16 per foot for ordinary 7 by 5 feet drifts. Sinking small shafts or winzes costs $24 to $35 per foot down to 100 feet.

The miners work but 8 hours a day, and are a very poor set of workmen. This is due partly to the climate, which is enervating, and partly to the idleness, which becomes habitual to men who have been employed simply to hold a mining lease. The government requires a certain number of men to be constantly employed on each claim to hold it.

The managers are fast replacing the old shirks with good men fresh from other camps.

Ore-Treatment.

A great number of experiments are at present being conducted on the Kalgoorlie ores. Nearly all known processes, and several never before heard of, have their advocates. Of course, some valuable knowledge will be gained by all this experimenting, and just as surely a great deal of very expensive machinery will in a short time be consigned to the scrap-pile.

The oxidized ores are treated with fair success by amalgation and blanket-concentration. It is the sulphide-ores that are puzzling the mine-managers.

The following is a screen-analysis of the pulp of a typical ore of the camp:

Out of 100 parts:

0.8	remain on a	24-mesh	screen.
0.98	"	30	"
9.34	"	40	"
16.63	"	60	"
17.43	"	80	"
5.50	"	100	"
49.32	passed through	100	"

And the following is a chemical analysis of a similar ore:

		Per cent.
	Silica,	48.43
	Iron,	10.24
	Alumina,	1.98
	Lime,	9.86
	Magnesia,	2.03
	Sulphur,	3.66
	Copper,	0.35
Calculated.	Carbonic acid,	7.75
Calculated.	Oxygen in oxide of iron,	3.05
Calculated.	Alkalies, etc., undetermined,	12.65
		100.00

From the above analyses two things are evident: first, that the 50 per cent. of slimes will be the great cause of trouble in any leaching-process; secondly, that the 10 per cent. of lime places chlorination out of the question.

At present the fine slimes are separated by partial-settling apparatus, such as the Butters rotary settler; the settlings being cyanided and the suspended slimes being settled later, and banked for subsequent treatment. The same general method is employed in the dry-crushing mills; the separation in that case being done with an air-draft.

As yet the finer slimes have not been successfully treated on a large scale; but some one of the ingenious adaptations of the agitation or filter-press processes, now in the experimental stage, will undoubtedly solve the problem.

I fancy that the treatment of these fine slimes will always be an expensive matter.

At present most of the unoxidized ore is shipped to the smelters in the Eastern colonies (see table of costs), because of the lack of roasting-apparatus.

Besides the question of ore-treatment, which is only a mat-

ter of study and experiment, there are two other serious problems facing the managers of Kalgoorlie. One is the water-difficulty; the other, the question of fuel. At present there is no lack of either. The native bush furnishes fire-wood which for calorific power has few equals. One cord of bush-wood closely piled will produce as much steam as one ton of good coal.* But wood is expensive now, costing $5 per cord, and the fuel-bill is from $1.50 to $2 per ton of ore crushed. There are very few cords of this wood to the acre, and it is only a question of time when the country will be denuded for miles in all directions from Kalgoorlie.

A railroad to Esperance would cheapen the price of coal, which sells at present for $11 per ton. There are coal-fields on the Collie and Irwin rivers; but as yet the deposits have not been proved to have commercial value.

The water at Kalgoorlie comes mostly from the mines. It is quite salt, and must be condensed for most purposes. Fig. 6 shows a plant of condensers. The water-supply is gradually becoming smaller as the veins are pumped out, and at the same time the demand is increasing. The escape from a water-famine lies in extending the pumping-area, and in economizing the water used in milling; and this is what is being done at the present time. Pumping-plants have been erected at several mines which have not proved valuable as ore producers, and the water is pumped to the producing mines. It is claimed that milling and cyaniding can be done with the loss of 0.25 ton of water to the ton of ore, and cyaniding alone with a loss of 15 per cent. the weight of the ore in water. Of course the water is used over and over again. At present, in most of the mills, about 0.5 ton of water is lost per ton of ore.

Discussion of the water-problem would be incomplete without mention of the great Coolgardie water-scheme. The plan is to collect the water of several streams in the Darling range, pump it 300 miles, and elevate it 1000 feet to a reservoir on the top of a hill called Mt. Burgess. From Mt. Burgess the water would be distributed by gravity to Kalgoorlie, Coolgardie, White Feather, Menzies, and all camps in the vicinity. The

* The manager of the Coolgardie Electric Light Co. gave me this information. Both coal and wood are used at the Company's plant.

estimated cost is $12,500,000. Mr. A. Bargigli, the engineer for a French syndicate, told me in London that his company has subscribed the funds, and that this large piece of work is to be undertaken at once.

As to the future output of the camp, I cannot do better than to give the average of estimates made by six well-known managers there. According to their estimates, the output for 1899 will be about $16,500,000.

I am much indebted to Mr. Edward Skewes, Manager of the Boulder Main Reef; Mr. Hamilton, Manager of the Great Boulder; and Prof. J. F. Kemp, for courteous aid in collecting the information contained in this paper.

The following table of the costs of ordinary commodities will give an idea of the expense of mining in Kalgoorlie.

Kalgoorlie Prices.

Wood, wholesale,	$5 to $6 per cord.
Coal, "	11.00 per ton.
Charcoal, "	0.60 per sack.
Fresh water, wholesale,	1.75 " 100 gallons.
Coke, wholesale,	25.00 " ton.
Mine timber, sawn, wholesale,	6.25 " 100 feet super.
" " round, "	0.11 " linear foot.
Nails, 3-inch wire, "	6.25 " 100 pounds.
Car-rails, wholesale,	48.00 " ton and upwards.
Drifting,	11 to 16 per foot.
Sinking,	24 to 35 " "
Beef, retail,	0.22 to 0.30 per pound.
Flour, "	4.75 per 100 pounds.
Butter, "	0.60 " pound.
Eggs, "	0.60 " dozen.
Bacon, "	0.25 " pound.
Blacksmith's coal,	18.80 " ton.
Miner's wages,	2.50 to $3.50 per day.
Freight to Fremantle on ore, large shipments,	4.00 per 2240 pounds.
Freight to Fremantle on ore, small shipments,	4.40 " " "
Wharfage,	0.50 " " "
Steamer-freight to Dry Creek,	3.00 per 2240 pounds.
Smelting at Dry Creek (95 per cent. of gold paid for),	12.20 " " "
Total for smelting at Dry Creek,	20.10 " " "
Steamer-freight to Port Pirie,	1.75 " " "
Smelting at Broken Hill (92 per cent. of gold paid for),	9.75 " " "

Notes on the Stockholm Exposition and the Iron and Steel Trade of Sweden.

BY JAMES DOUGLAS, NEW YORK CITY.

(Atlantic City Meeting, February, 1898.)

I HAD the good fortune to visit the Stockholm Exposition just before its close in October last, and to get a glimpse of the methods used in Sweden in making the wonderful steel and iron for which its furnaces are so famous. My time and opportunities for observation were limited, but some points which I noted may be worth bringing to the notice of the Institute.

The Exposition itself was confined to the natural and artificial products of Scandinavia, and therefore iron and steel held a very prominent place in it. There was a complete collection of the rocks and ores of the Peninsula made by the Geological Survey; and every iron-mine of note exhibited a glass model of its workings, as well as classified specimens, accompanied by thorough analyses of the ores. Another feature of interest was the display of models of the old mining-machinery, in the invention and introduction of which Sweden led the way, and of the old and new surveying-instruments and compasses devised for the exploration of the Swedish iron-ore bodies, all of which are more or less magnetic. The magnetometric collection and exhibit of methods of operation made by Prof. Nordeström were supremely instructive. The larger metallurgical works, such as those of the Kopparberg and Sandvik companies, exhibited their wares in separate pavilions, in which crude and manufactured objects were arranged most effectively and tastefully; while the smaller companies erected in the Industrial Arts Building trophies, every one of which was an object of beauty. One was struck on all sides by the fine æsthetic feeling exhibited in the ingenious installation of materials and objects not easily lending themselves to artistic treatment. Hammers, picks and drill-steel were grouped into mural designs which it was a delight to look at.

The most interesting exhibit was that of the Kopparberg Co., a concern which traces back its active history to the thirteenth century. For hundreds of years its operations were devoted almost exclusively to the mining and smelting of copper from its great Falun mines. From carefully-kept records it is estimated that this property has yielded in all about 500,000 tons of copper, 15 tons of silver, and 1.25 tons of gold. At the present time, however, the copper-operations of the company are extremely limited, the ore in depth yielding only from 2 to 3 per cent. of copper, in a compact iron pyrites, which is used for acid-making. The report of the company shows that it produced in 1896 about 400 tons of copper, 1600 tons of bluestone, 2300 tons of acid, and some gold and silver.

One of the exhibits of the company was a copper coin of 1644, a 10-daler piece, the nominal value of which in our money is about 80 cents. It is a sheet of coined copper about 18 by 18 inches square. I have a half-daler piece, about 3 by 3 inches, dated 1749. It weighs about 13½ ounces, and its nominal value is about four cents. Until about the middle of the last century, the Swedish government strove to make a market for its copper production in the same way as our bimetallists to-day wish to market our silver, namely, as coin.

The Kopparberg exhibit illustrated also the subsidiary industries which have grown up around most of the prominent mines of Sweden. Not only does the same company mine the ore and reduce it to metal, but it converts the metal into highly specialized forms, and thus combines manufacturing with mining and smelting.

Besides such industries, lumbering and the making of wood-pulp and paper have become auxiliaries to several of the iron-works. As coal or coke, being more or less tainted with sulphur, is scrupulously excluded from their iron-works, charcoal is exclusively used in the blast-furnaces and wood producer-gas in the open-hearth furnaces. Enormous quantities of wood are therefore consumed as fuel. But as inferior lumber is suitable for charcoal and the refuse of the lumbering-camp and of the saw-mill can be converted into gas, it has been found profitable to sort the lumber into three grades, the best going to the pulp-mill, the second quality being sawn into lumber, and the poorer grades only being used as fuel.

How diversified are these several industries of the iron companies may be judged by the list and quality of articles made by the Kopparberg company, though it must be remembered that the operations of this company are by far the most extensive in Sweden. In addition to the copper produced, and the by-products, some, but not all, of which are enumerated above, the company, in 1896, made 55,000 tons of pig-iron, 35,000 tons of Bessemer ingots, 26,000 tons of open-hearth steel, 4000 tons of charcoal-blooms, 46,000 tons of rolled and hammered iron and sheets of all sizes, 1000 tons of horse-shoe nails, bolts, nuts, spikes, etc., 5500 tons of wood-pulp and 57,369 St. Pet. stds.* sawn lumber. The company's operations extend over three provinces. They own 736,000 acres of forest-land, which is traversed by 1700 miles of lake and water-courses. It would seem, therefore, that they cut over about 5 acres to the ton of product.

At the Munkfors works, where I was courteously received, I understood that an area of 200,000 acres supplied first class wood for pulp, second class for lumber and third class for the manufacture of about 30,000 tons of product, which would be at the rate of about 7 acres per ton of finished iron and steel. There, also, part of the iron left the works as pig, and part in highly finished forms of steel and iron.

The Crown forests, which cover some 8,000,000 acres, or one-sixth of the forest-land of Sweden, are scientifically cut over and preserved, and the forest-domains of the large iron companies are also scrupulously husbanded, only the increase being cut. A very large tree is seldom seen in Sweden, as it is found most economical to cut spruce and pine after forty years' growth, when the butt has attained a thickness of from 8 to 10 inches.

At Kopparberg, the charcoal is burned in a continuous kiln at a cost of 33 per cent. less than that of heap-burning, and with 22 per cent. higher volume of yield; but the wasteful charcoal-burning in heaps is elsewhere generally, if not everywhere, practiced.

The sawdust from the mills, bark, twigs, and every particle

* The St. Petersburg standard is 165 English cubic feet, and therefore 1980 feet board-measure.

of combustible vegetable matter, is made into gas in producers.* These are large brick chambers, the floors of which are funnel-shaped. The fuel is fed through hoppers in the roof. Coarse fuel is burnt in a layer about 12 feet in depth; but when sawdust is being turned into gas, the depth is only 6 feet, and a light forced draft is used. Where the producers are within 50 feet of the open-hearth furnaces, the tar and water are not condensed, and perfect combustion in that case is secured only by the employment of Siemens regenerators; but when the producers are at greater distance the tar and water are eliminated in condensers which do not differ greatly from those described as the Lundin condenser by Mr. A. S. Hewitt in his famous Paris Exposition report of 1867.

The purer iron-ores, with phosphorus as low as 0.0045 per cent. are those which maintain Sweden's reputation; but though most of the highly phosphoric ores of the province of Kopparberg are exported (namely, in 1896, 550,000 tons out of a total output of 639,000 tons), the balance is made into very superior iron by the Thomas-Gilchrist process with wood-fuel.

The Swedish iron-master prefers, however, to restrict his operations to the production of small quantities of highest-grade material rather than to aim at a large output of inferior iron and steel. The national statistics clearly demonstrate this.

Mr. Brough (*Journal of the Society of Arts*, December 10, 1897) gives the following for the production of 1896, a year later than the official statistics which I possess. He says:

"There were 124 works at which an aggregate of 140 blast-furnaces were in operation, and the production of pig-iron comprised:

	Tons.
Forge pig-iron,	246,022
Bessemer and open-hearth pig,	225,103
Spiegeleisen,	738
Foundry pig,	15,284
Direct castings,	7,271
Total,	494,418

"The blast-furnaces produced on an average 3,532 tons per furnace during the year. The time of production for each furnace was 271 days, and the mean daily

* For drawings of types of such producers, see Mr. Odelstjerna's paper on "The Manufacture of Open-Hearth Steel in Sweden." *Trans.*, xxiv., 288.

In addition to the producers described by Mr. Odelstjerna, producers without grates are used for gasifying wood, both in sticks and in saw-dust.

out-turn per furnace was 13.03 tons. Charcoal is practically the only fuel used in the Swedish blast-furnaces, for it is only in very rare cases, as in the manufacture of spiegeleisen, that a little English coke is used.

"There were 137 wrought iron and steel works, with 353 finery hearths, 4 puddling-furnaces, 29 Bessemer converters, 41 open-hearth furnaces and 4 crucible-steel furnaces. The production comprised:

	Tons.
Wrought iron,	188,396
Bessemer ingots,	114,120
Open-hearth ingots,	142,301
Crucible-steel ingots,	604"

With the exception of the Kopparberg, and perhaps one other company, none of the works, I believe, turn out more than 30,000 tons of all kinds of iron and steel. Even the famous Sandvik Co., whose steel tubes are celebrated in this country for bicycle-frames, makes only about 20,000 tons of steel annually. But the utmost care and supervision is exercised, and hand-labor is employed to an extent which is profitable only when expended upon material of such value. Every wire bar is carefully inspected, and men with hammer and chisel cut off every particle of scale or burnt surface. The Swedish steel is too valuable to be turned even into rails, for the rails in the very yards of the steel-works are generally of Dowlais manufacture.

While, however, the iron- and steel-making and manufacturing from pure ores is conducted on what we would consider a very limited scale, the mining and exportation of ores of impure quality, though of high percentage, is assuming very large proportions. I have already referred to the exportation of over half a million tons of iron-ore from the province of Kopparberg, an export-industry which has grown up within 15 years, and will undoubtedly continue to increase. But the largest exportation will be from the mines in the northern provinces, which are as yet almost uninhabited, but which are known to conceal enormous deposits of iron-ore of non-Bessemer grade. At the Exhibition there were striking exhibits from the Gellivare group of mines, on deposits which vie in size with the largest of our Lake Superior iron-ore masses. In another respect they resemble some of our Lake Superior mines, in that the ores can be classified into grades with variable amounts of phosphorus, though no high-grade Bessemer ore is pro-

duced, the grades varying from 68 per cent. of iron and 0.05 of phosphorus to 60 per cent. of iron and 1.5 of phosphorus. As yet the ores have been shipped from the port of Lulea, on the Baltic, and have found their most accessible market in Germany. (At Stettin very large iron- and steel-works have been recently *erected to be fed entirely with imported ores.)* At Lulea the company has built a concentrating-plant of the capacity of 100,000 tons annually, with the double purpose of making a high-grade magnetite and separating the apatite as a valuable by-product.* A wider market, however, is being sought for the ores, and to this end a railroad has been projected across the neck of the peninsula to the coast of Norway, where the ports are open the year round, and where large works will be erected. The railroad will tap other large deposits of ore in this hyperborean region (67° to 68° N. lat.), and therefore Sweden may be counted on as likely to become, in the near future, the largest producer of iron-ore in the world, next, of course, to ourselves. Her production of ore in 1896 was 2,038,096 tons, and her exportation from the Kopparberg and Gellivare mines alone reached 1,175,000 tons.

Of all the wonderful objects exhibited by the many companies which displayed their wares in Stockholm, the most striking were the cold- and hot-rolled bands made by the Sandvik works. These works are historically interesting as those in which Mr. G. F. Göransson was able to make good steel by the pneumatic method without the addition of spiegeleisen, though equal success on the lines laid down originally by Mr. Bessemer had not rewarded his efforts when working with the impure ores of England. These works, like others in Sweden, have accommodated themselves to the shifting demands of trade, and as one article after another, which, for a time, was profitably manufactured, declined in value, through disuse or foreign competition, they have found some other article to manufacture, for which their iron and steel are peculiarly fitted. At present the Sandvik works supply the world with a very large portion of its "Paragon" umbrella-ribs; but their ribbons of steel illustrate more expressively than do their umbrella-ribs what can be done with pure steel. In the Sandvik pavilion a band

* See Brough's paper in the *Journal of the Society of Arts*, December 10, 1897.

of cold-rolled steel of No. 26 gauge, 2354 feet in length and 8 inches in width, weighing 1146 pounds, was coiled like a frame around the objects displayed upon the walls, and hung in graceful festoons from the roof. There was also exhibited a band of No. 49 gauge, weighing 43 pounds, 2⅝ inches wide, 4317 feet long, and which was, therefore, about 0.00125 inch in thickness. The same band-saw which attracted attention in Chicago (No. 14 gauge, 220 feet in length, weighing 677 pounds) was exhibited, and they showed a strip for lap-welded tubing of No. 6 gauge, 21 inches wide, but with perfectly true square edges. This company also exhibited with pride a hot-rolled band of No. 8 gauge, 8 inches wide and 298 feet long, weighing 2140 pounds. This they claim is the heaviest and longest band rolled in one heat of such width and gauge from a bloom 4⅝ inches thick and 11 feet long.

A prominent place in their pavilion was given to their hollow blooms for the manufacture of bicycle-tubes, samples of which, up to 8 inches in diameter, were displayed with excellent effect.

Perhaps the most startling exhibit was that made by the Laval Steam Turbine Co., whose engines are now built with much more skill and operated more economically than when they made their modest *début* at Chicago. At their manufactory at Jerla, near Stockholm, I saw their engines running under a pressure of 225 atmospheres, though at Stockholm they did not venture to raise the steam quite to that high pressure. The engines are certainly models of exquisite workmanship. As at present sold, they are run under a steam-pressure of 180 pounds of dry steam, but within two or three years the company expects to have so perfected its high-pressure boiler as to warrant its being offered to the public with assurance of safety. The boiler consists, I believe, of a 1-inch coil, in which a very small quantity of water is evaporated under this enormous pressure, the condensed steam after use being always returned to the boiler. If a coil should burst, the steam, which is small in quantity, expends its energy innocently within the ample shell which encloses the coil. The engines and boilers, as now sold, are said to work with an expenditure of 2 pounds of good coal per horse-power, but under extreme pressure this consumption is said to be reduced to 1.5 pounds.

Notes on the Vein-Formation and Mining of Gilpin County, Colorado.

BY FORBES RICKARD, F.G.S., CENTRAL CITY, COLO.

(Atlantic City Meeting, February, 1898.)

GILPIN COUNTY, the cradle of mining in Colorado and the Cornwall of North America, is too well known to need much introduction; yet, for the benefit of those not familiar with the district, it may be well to premise that it comprises a tract of country of about 122 square miles, lying between the counties of Jefferson, Boulder and Clear Creek, and bounded on the west by the main range of the Colorado Rockies. It is located centrally in the State, 40 miles west from the city of Denver.

During the 38 years since the first gold-discovery, Gilpin county has contributed about eighty million dollars in gold and silver to the world's supply of these metals. Many mining districts have had their rise and decline in half this time; but, very far from being exhausted, this mining district is still in the ascendant; its output is increasing (it was $3,600,000 for the year just closed), and it was never more prosperous nor productive than to-day.

Taken generally, there does not exist anywhere in the world so great a number of permanent veins within so small an area as in this district.

Gilpin is practically a "poor man's camp." Its veins, with few exceptions, outcrop at the surface. In the earlier days of the camp these outcrops were worked in many instances by sluice-mining; particularly is this the case with regard to the veins of the Russell gulch.

Discoveries are still made in the heart of this district, and the productive area is every year extending its limits. The districts of Yankee Hill, Hawkeye and Pine Creek, which used to be isolated, are now part and portion of the one gold-field of Gilpin county.

Within the last year sensationally rich gold-discoveries have

been made in the country contiguous to the mineral lands (the Hall and Root ranches), held under old agricultural patents, which hitherto had been considered a distinctly silver district. Such discoveries, perhaps more than anything else, attest the possibilities of this, the oldest mining district of Colorado.

It is to be regretted that the active and useful work of the United States Geological Survey, resulting within recent years in elaborate reports on the ore-deposits of Leadville, Aspen and Cripple Creek, has not included old Gilpin county. In the instance of Cripple Creek particularly, the work of the Survey has been invaluable to the rapid and successful development of this famous camp. There is certainly much in the vein-formation of this district of Gilpin which offers an inviting and interesting field for investigation.

The veins of this county occur in a metamorphic area, *i.e.*, an area which has been subjected, through heat, pressure and other agencies, to physical and chemical changes so great that the rock-mass has retained little of its original character. To the varying proportion of the quartz, feldspar, mica and other component minerals, and to textural variations in the rock-mass, is due an endless variety of the rock-species of the gneiss-granite order, granitite, protogine-granite, granulite, felsite and pegmatite being among the more important.

The gradation from one type to another is almost imperceptible; a metamorphic granite passes into gneiss, and this again into rock of schistose character. The prevailing schist is micaceous, talc and hornblende schists occurring in a minor degree.

The country everywhere shows distinct bedding, dipping easterly.

These rocks frequently afford examples of the segregation of mica, quartz or feldspar out of the main rock composed of all three minerals. Sometimes the quartz and feldspar crystallize together distinctly from the mica. (See Fig. 1.)

In some parts of this district there are to be found patches where the component minerals of the gneiss-granite are very coarsely crystalline, the mica occurring in plates several inches in diameter, with the quartz and feldspar proportionately coarse.

Although it is still a question among geologists to what extent the older gneisses are, in general, to be considered as metamorphosed sediments, the gneiss of this district is, I think, commonly recognized as such; and with the exception of some irregular and inextensive occurrences of rock of the syenite and diorite order, which appear to be holocrystalline, there is, I think, no true granite in this whole mineral district.

The mining of this old camp has brought to light a great deal that bears upon the geology of mineral veins, and the light afforded by the last twenty years of observation and research on this subject as applied to mining in different parts of the world renders it possible to speak with more intelligence on vein-phenomena than previously. Still, geology in its bearing on the deposition of ores and the formation of veins leaves much to speculation and conjecture.

The vein type of this district is defined by the term "true fissure," and the fissure-vein is held to be the simplest kind of ore-deposit.

The lodes of Gilpin present no volcanic complex, such as that of Cripple Creek, where this feature renders the pursuit of the veins very difficult; nor have we here to deal with intricate planes of faulting, such as are characteristic of the formations of the Leadville belt, and have so much complicated mining in that locality. The fissures of Gilpin sometimes occupy a line of faulting, but the displacement seldom exceeds more than 2 or 3 feet. These veins usually have one well-defined wall, and sometimes both walls of the vein are very clearly marked; but a well-defined vein does not necessarily mean a productive vein; for the filling along some parts of these veins, though the vein itself may be wide and not lack definition, is often scarcely distinguishable from the material of the vein-walls. In the more productive parts of the main veins of the district one wall is usually much broken.

The district contains a gold-belt and a silver-belt, which are fairly distinct, though they verge very gradually into each other on the confines. The silver-belt occupies the eastern limit of the region about the town of Black Hawk, immediately east of North Clear Creek, crossing this creek just south of that town in a line approximately north and south. So long as silver commanded a good price, the veins containing that metal

were extensively worked; and although, in recent years, operations in the silver-area have been much restricted, silver, nevertheless, forms no unimportant portion of the values produced by the gold-mines. It is associated with the gold in varying degrees in all the ores, and forms a considerable element of value. The average proportion of silver to gold throughout this county is about 5 ounces of silver to every ounce of gold. It is worthy of note, that whereas such is the association of gold and silver in the gold-belt, the ores of the silver-belt proper carry little or no gold. The Coaley and Gilpin lodes, which are among the oldest patented claims in the State, carry no gold. A shipment of 73 tons was at one time made from the latter mine, which returned an average assay of 215 ounces of silver per ton, and no gold.

The location-survey of Gilpin shows two very distinct sets of veins occupying its mineral area. One set has a course approximately east and west, the other strikes northeast and southwest. The main features of the one group are shared by the other.

Intrusive dikes frequently occupy both these lines, occurring sometimes in conjunction with mineral veins. The material of these dikes has been determined to be andesite. It is, as a rule, much altered and silicified by reason of its proximity to ore-deposits.* The decomposition of the feldspar crystals (which are conspicuous against the finer-grained ground-mass) gives this dike-rock a mottled or porphyritic appearance, for which reason it is generally known to miners as porphyry.

This eruptive is found widely distributed throughout this section. (See Figs. 4, 5 and 6.) It varies in color from grayish-white to green, and sometimes purplish. In grain it gradates from a coarsely crystalline to a compact, close-grained rock. On Gregory hill, on the east side of Packard gulch, there is a fine exposure of this dike-rock in which very perfect orthoclase crystals from 0.5 to 1.3 inches long (sometimes twinned on the Carlsbad type) occur, imbedded in a fine-grained feldspathic base, in which minute crystals of augite and magnetite are visible.

* Prof. Judd, of the Royal School of Mines, London, determined the rock of the dike that accompanies the California vein to be a quartz-andesite, sometimes called dacite.

These differences of crystallization, as explained in other instances, are, no doubt, due to the effects of a slow or rapid cooling of the magma at the time of its consolidation. The dike of Gregory hill is more than 80 feet wide in places, and it is at its center that some of its component feldspar has attained such remarkable development. The interior of the dike would naturally retain its heat longer than its edges, which would be cooled by contact with the sides of the fissure, and thus favor a coarser crystallization towards the center.

Other igneous rocks, previously referred to, are rarely found within the most productive sections of this county's mining area.

In some parts of the district large, dike-like masses stand out prominently from the general rock-mass, which might, at first sight, be mistaken for dikes, but are nothing more than bars of harder rock than that enclosing them on either side. The distinction between the dike-like masses and the contiguous ground is not one of composition, but only a physical distinction. The softer rock on either side of these bars has been eroded, while the latter, being harder, have been better able to resist the weathering. (See Fig. 2.)

The andesite dikes appear to have a very important influence on the vein-system of this district. They frequently form a wall of the veins, and it is to be remarked, when that is the case, that the divergence of the vein from the dike will usually coincide with an impoverishment.

The dikes vary in width from 6 to 30 feet. That they are of earlier origin than the mineral veins is shown by the many instances where the latter are found to cross them.

The common type of lode-filling, or vein-stuff, consists of a smelting-ore, or "iron-streak," and a proportionately larger width of milling-ore. It sometimes carries, on either the hanging- or the foot-wall, a selvage or clay-seam, and, at other times, is without much parting. The smelting-ore is the more compact ore, usually iron pyrites, with a sprinkling of copper pyrites, and the milling-ore consists, for the most part, of feldspathic matter and some quartz, impregnated and seamed with the same sulphides; in color it is white to greenish. The word "smelting" is here applied with reference to its local use. As all ores, nowadays, are more or less amenable to smelting, the

FIG. 1.

Segregation of Mica from the Gneiss-Granite Rock-Mass.

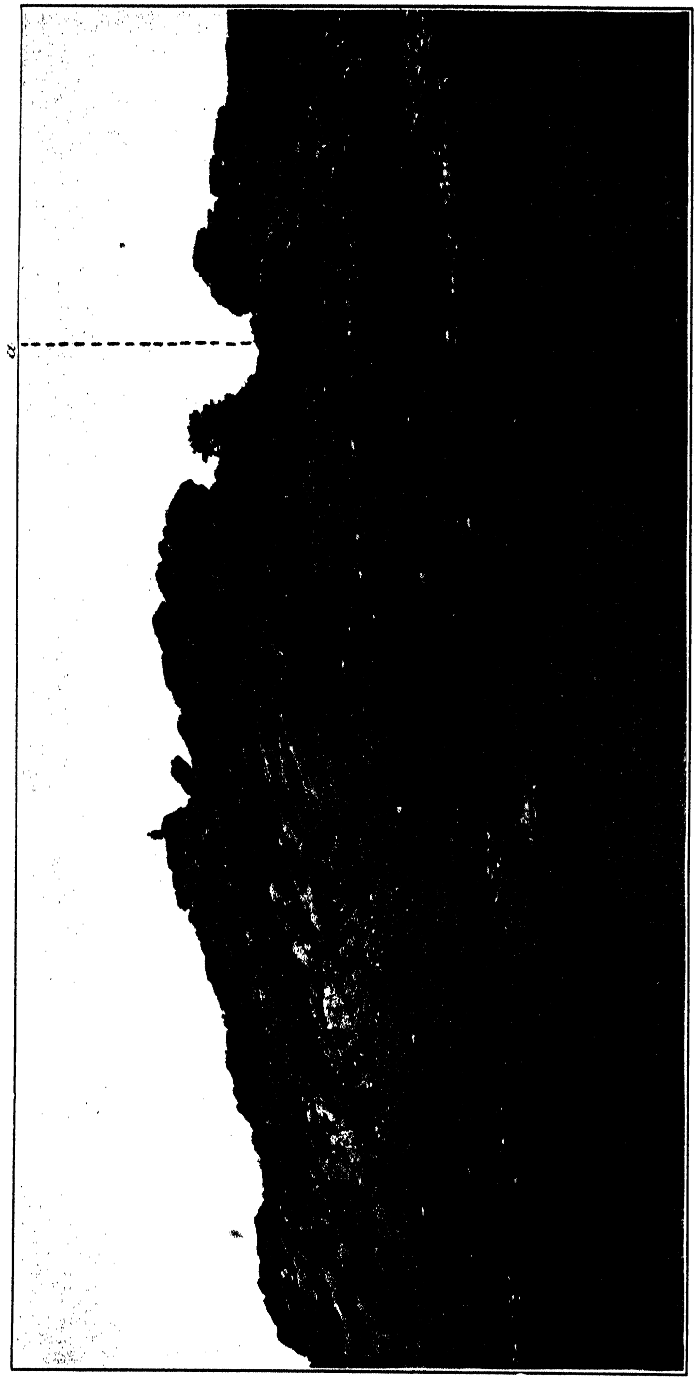
Fig. 2.

Dike-like Bars of Gneiss-Granite.

Fig. 3.

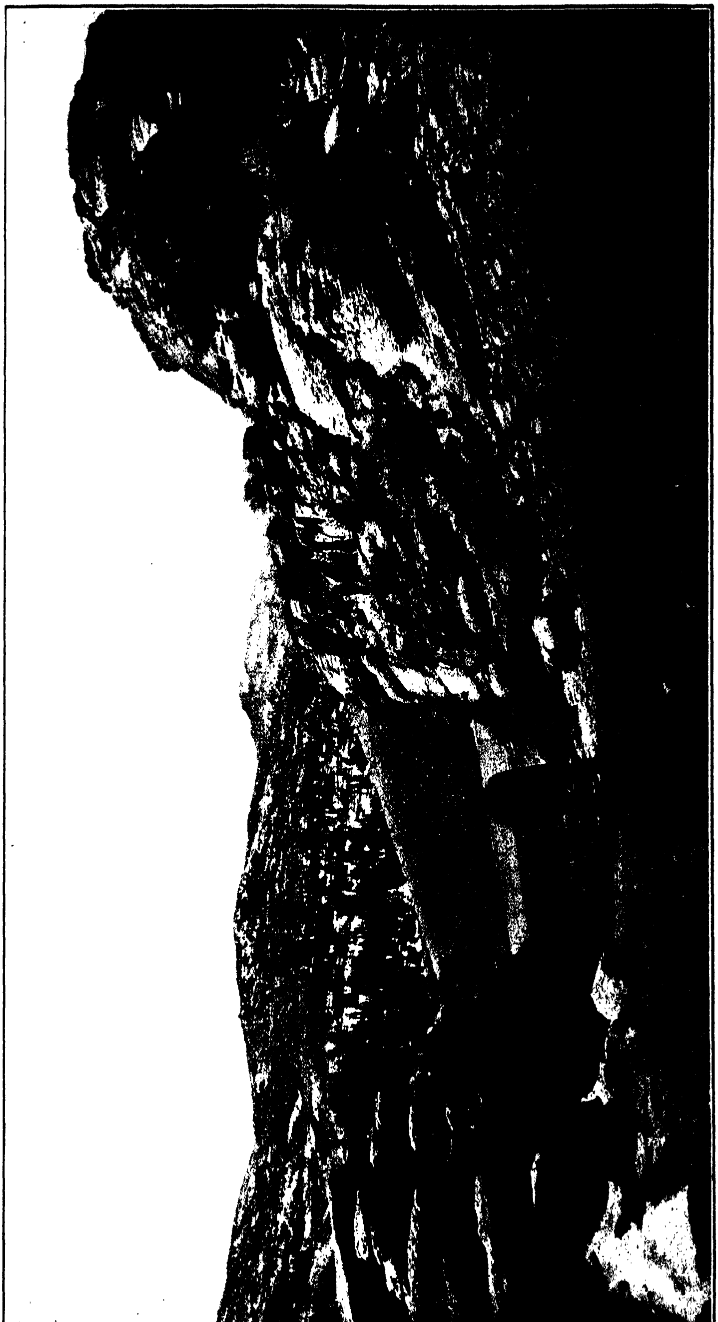
Granulite Outlier, Showing Tilting of Strata Vertically. (For proportions, note figure in foreground.)

FIG. 4.



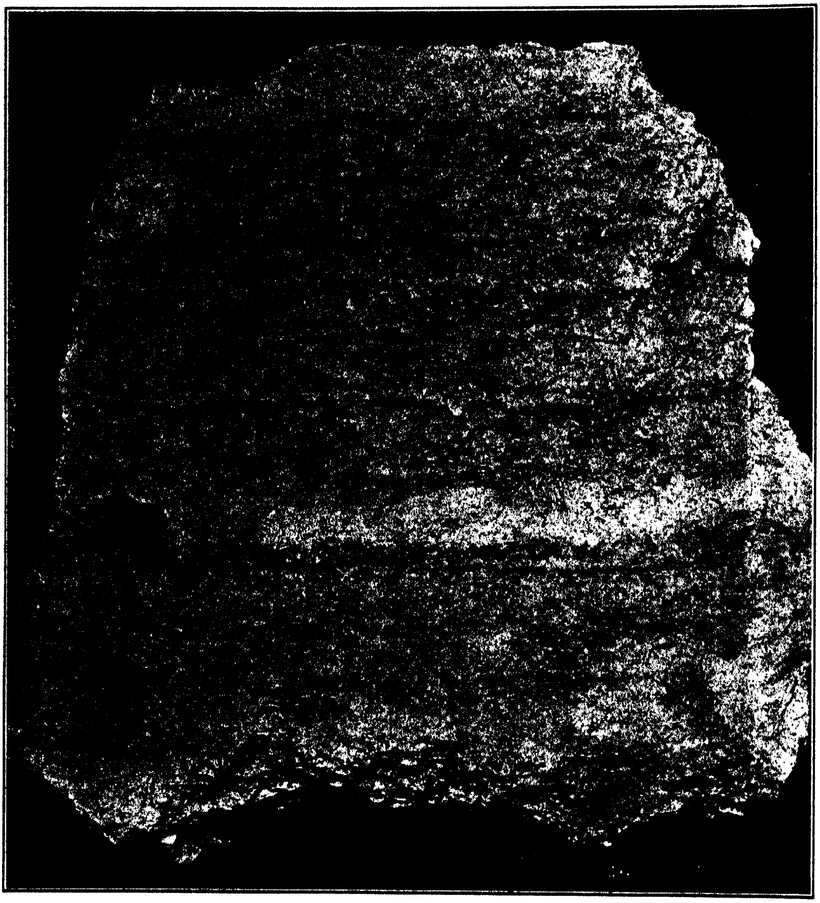
The "Queen's Chair," a Peculiar Exposure of the Andesite Eruptive. (The west side exhibits a fault, which has created a hollow space in the center of the rock-mass. For a view at *a*, see Fig. 5.)

Fig. 5.



The Reverse Side of the Formation Shown at Point *a*, Fig. 4.

FIG. 6.



Rock of the Formation of Figs. 4 and 5, Showing Banded Structure (¼ natural size).

grade of the ore (*i.e.*, the per-ton value of the ore), more than anything else, determines its suitability for treatment by mill or furnace. The proportion of smelting- to milling-ore taken over the whole county would, I judge, be approximately 5 tons of smelting-ore to every 100 tons of milling-ore.

Among the ore-varieties, pyrite is the principal gold-carrier of the camp, and is commonly associated with chalcopyrite, zinc-blende (the dark ferruginous and yellow resinous varieties), galena, and gray copper (*fahlerz*), the latter in two varieties, the common tetrahedrite and an arsenical (red-streaked) variety (tennantite ?). Carbonate ores, such as chalybite, cerussite, malachite and rhodochrosite are also found with these ores, but only in relatively small proportion. Enargite has been mined in some parts of the gold-belt.

Tellurium has been found in association with the ores of many of the mines of this district, and in very appreciable quantities. It seems to play an important part as a gold-carrier, and occurs more commonly associated with the ores than is generally supposed.

Dr. Richard Pearce, in a paper read before the Institute several years ago,* writes of "the wholesale distribution of tellurium in masses of pyrite" in the Leadville sulphides, where the existence of tellurium sulphide was never previously suspected. In this instance the tellurium was discovered in the process of kiln-roasting. Tellurium in small quantities has also been detected by Dr. Pearce in ores of this camp that have been treated at the Argo smelter, the presence of tellurium manifesting itself during the process of calcination.

Bismuth and arsenic are found very appreciably associated with ores from this district, and it is to their presence that has been attributed some of the trouble experienced in the earlier days of smelting in this camp. Bismuth usually occurs with the gray-copper (*fahlerz*) ores.

Arsenical pyrites (mispickel) is quite rare, but arsenic in very small proportion is present in nearly all of the iron sulphide ores of Gilpin.

Uranium (pitch-blende) has been found in considerable quantities in this district, notably in the Wood and Kirk mines of

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* *Trans.*, xviii., 451.

Leavenworth gulch. In early days a shipment of several tons of this uranium-ore, taken from the Wood mine, was made to a smelting-establishment in Swansea, England, where it brought a large price. The market for this comparatively rare mineral has, in recent years, been exceedingly limited, but uranium is again in demand for some patented use in connection with the manufacture of armor-plate, and an improving market has given a fresh impetus to the mining of pitch-blende in this district.

Barite and calcite, so common in limestone districts, are here of rare occurrence. The rarity of calcite is not improbably due to the absence of the plagioclase lime feldspars in the rocks of this formation.

Polybasite, stephanite and horn-silver are found in the silver-belt of the area.

Titaniferous iron, epidote, magnetite, garnet (almandine), tourmaline (more commonly Schorl) and graphite are of common occurrence in the country-rock enclosing these veins.

Some remarkably fine ore-specimens have been taken from the mines of this county, exhibiting metallic gold in several forms, and notably as an incrustation on pyrite (in the St. Louis Gunnell mine), as leaf-gold, in a crystalline form as octehedra, and as a pseudomorph after pyrite (in the Gregory mine). Very little, however, of the gold of these ores is free in the gangue-rock. It occurs almost altogether in mechanical association with the sulphides of the ore.

There are parts of the mining area of Gilpin where the ores of iron, copper, copper-antimony and zinc predominate very distinctly over any of the other metallic sulphides; so much so, in fact, as to make any particular ore typical of a locality.

This distinct prevalence of a particular ore-species might at first suggest differences of origin for the several ores, but there is nothing in that idea. For, however much any of these ores may predominate in any particular instance, the main veins of the county contain, in varying proportions, all these ores; and I therefore fail to see grounds for attempting to prove any order of paragenesis of the metallic elements of these lodes.

In some of the ores of the district it is a common thing to

find the sulphides of iron, copper, lead, zinc and antimony occurring in intimate mixture, and by testing the sulphides of any complex ore I have found that, in general, the copper pyrites carries most of the gold; the lead, zinc and antimonial sulphides carry the greater part of the silver-contents; and the iron pyrites will assay very close to the average value of the whole ore. Coarsely-crystalline pyrite is usually of low grade.*

The ores of the silver-belt are galena, zinc-blende and pyrite, and occurring with them are found, in subordinate quantity, the rarer silver minerals to which I have previously referred.

The gangue of the silver-veins does not materially differ from that of the gold-veins, and in appearance the commoner silver ores are hardly distinguishable from the gold-ores of this camp. The country-rock in both cases is the same.

To the question, so often uppermost among miners, what is going to be the influence of depth on the lodes of this remarkable old camp, there can be no reply but an extremely hazardous one. Here, as in other parts of the world, where gold is mined from granite and granitoid-gneiss, there is no rule to guide us in such speculation. The Gilpin miner may not hope to go on indefinitely in the pursuit of his lodes in depth; there are, alas! too many abandoned "prospects" which evidence the limit of the ore that some luckless miner was following; and still he may well take courage from the fact that experience in the mining of the main fissure-veins of this district has inspired faith in the outcome of explorations to considerable depths.

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* Mr. T. A. Rickard in his book on *The Stamp-Milling of Gold-Ores* (p. 26) gives the following result of an analysis of a typical piece of ore from the California mine, which is interesting in this connection:

	Gold. oz. per ton.	Silver. oz. per ton.
Iron pyrites,	0.65	4.85
Copper pyrites,	0.85	53.50
Gray copper,	0.90	38.65
Blende,	0.16	6.45
White quartz,	3.32	7.35
Bluish quartz,	3.56	5.84
Flinty quartz,	0.18	1.90
Feldspathic gangue,	0.90	2.35

Not improbably, Gilpin is destined to be the seat of deep mining, though at present its mines are seldom deeper than 1000 feet, and there is only one case of a vertical depth of 2000 feet. This is the case of the California mine, where a shaft has been sunk on the lode to a depth of 2200 feet, dipping at an angle of 85°.

The lodes of Gilpin show oxidation to a depth varying from 40 to 80 feet, and in some cases exceeding 200 feet. The Carr mine is a good instance of deep oxidation.

Once the workings have got beneath the altered region, depth brings no perceptible change of value to the vein or to the character of the bullion recovered from its ores in mill or furnace.

In the deepest mining, that of the California mine just instanced, the lode shows no essential change of character in its deepest developments; and although sinking on that vein has been abandoned for some years, my familiarity with its lowest levels prompts me to say that had it not been that the main working-shaft of this mine was several hundred feet away from the main ore-chute of its workings, this mine would have been working to-day at good profit in still lower depths. Work on this vein is now almost wholly confined to its western extension, where it has been in *bonanza* for the last three years, at a depth ranging from 600 to 1000 feet.

In writing of the fissure-vein, Mr. Emmons says: "It has long been the opinion of the writer that the idea generally accepted among miners, that because an ore-deposit has been formed on what may be called a 'true fissure-vein,' it necessarily has an indefinite extension in depth, is much in the nature of a popular fallacy, and that the extension in depth of ore in a fissure is as likely to terminate within a measureable distance as the extent of ore-deposition on what are generally called 'blanket deposits.'"

It is safe to say that the "measurable distance" in the main veins of Gilpin county has yet to be sounded at some point deeper than hitherto reached by its deepest explorations.

The lodes of Gilpin, of which the California has been cited as a good type, hold their values with remarkable persistency; and it is a noteworthy fact that the lodes which yielded well near the surface are, generally speaking, those which are prof-

itably pursued in depth; and further, that certain low-grade pyritic veins which yielded poorly at their outcrops are persistently low-grade producers in their present deeper developments.

When, at a comparatively shallow depth, some of these veins have pinched, or when the grade has fallen off, it has been the habit among miners of this district to use the word "cap" as entirely explanatory of these conditions. The expression is senseless, for all these veins, even the richest, have and have had their barren intervals, though, on the whole, they are persistent. Some veins are poorer than others, and a "cap" will not disguise the fact.

There are no "blanket deposits" in this district. The veins cut obliquely across the foliation of the country-rock. Occasionally ore will be found occupying the line of lamination of the enclosing gneiss or schist, but I know of no instance where ore occurring in this way has shown any degree of permanence; nor are very flat veins found to be productive in this section. The vein of the Specie Payment mine is one of the rare exceptions.

Among the properties of the vein on which the ore in these lodes is, to a greater or less extent, dependent, are its direction of strike and dip. The variations of value observable with dip are distinctly suggestive of some relationship between it and the vein. Instances are frequently to be observed where a change in the dip of the vein has brought about a change in the character of the vein, sometimes bringing enrichment and at other times impoverishment. In my observation of these veins I have repeatedly noted that a departure from the normal dip of the vein towards a steeper dip coincided with a change in the vein for the poorer. Again, the dip seems to have some peculiar relation to the topographical structure of the immediate country, which it would be interesting to work out. I have remarked that where the main producing veins of this district are situated on the hillside they almost invariably dip into the hill. There are, however, some exceptions.

The junction of veins, either of which carries paying values, is, in this district, as in many other districts, regarded as likely to result in a local enrichment or good paying ore-body at or about such junction. There is no doubt that many of the

most productive ore-chutes of the mines of Gilpin county are traceable to the meeting of two or more veins; but it has been my observation that frequently, where the strike of two veins seemed to make the likelihood of a junction very certain, there has only been a convergence of the veins, which, after running nearly parallel for a short distance, diverged again. The rock intervening at the point of convergence has sometimes been mineralized to such an extent as to give it a paying value without there being any actual junction of the veins themselves; at other times it is barren. A good example of the former occurrence is to be seen in the First Centennial vein at about 375 feet from the surface.

Though the veins of this district exist, for the greater part, independent of eruptives, still, in some cases, they occur on the contact between eruptive dikes and the metamorphic rocks. In such instances the vein is often peculiarly dependent on the dike.

The irregular enrichment, in point of size and value of veins, under apparently similar conditions for ore-deposition, is perhaps, in a great measure, to be accounted for by the generally recognized fact that the physical texture of the rock, as well as its chemical composition (which is always a factor in vein phenomena), exerts an important influence in the precipitation of mineral from solution. It is sometimes explained by some theory of secondary deposition, which did not reach over the whole vein, but are not all ores the result of repeated or many times repeated deposition?

Sometimes a vein will ramify into a dike so as to scatter all through the porphyry mass, and when the vein is rich, it may give to the whole an economic value. In the neighborhood of the Climax and San Juan mines, on Quartz hill, there is an area known as the "patch," which is impregnated and seamed with ore to such an extent as to give the whole mass an appreciable value in gold and silver. This is a pear-shaped area, occupying 10 to 15 acres at the surface and narrowing with depth, but proved in the San Juan mine to extend beyond a depth of 800 feet from surface. It is generally and erroneously referred to as a porphyry overflow. The andesite eruptive forms the smaller part of this mass, and occurs within it in irregular and disconnected bodies. The main body of the

"patch" consists of a crystalline-granular mixture of quartz and orthoclase, with little or no mica. The feldspar is partly kaolinized, and the rock contains much drusy quartz. Through the mass are sparsely disseminated pyrite, zinc-blende, chalybite, and, to a lesser degree, galena and chalcopyrite.

I would suggest that the patch is nothing more than the altered gneiss-granite country, which here occupied an area where the agency of percolating waters was very active in promoting its decomposition and alteration. Another interesting feature of this "patch" is that it is traversed in parts by streaks of high-grade pyritic ore, varying in thickness from knife-blade seams to 2 or 3 inches, and assaying from 5 to 15 ounces of gold per ton.

The San Juan mine is peculiar, too, on account of large rounded boulders that have been found in its vein several hundred feet from the surface. The occurrence of these boulders, varying in size from a few inches to three feet in diameter, has been accounted for by many a puzzled observer by imagining the fissure to have gaped at some time in its history, and so permitted the falling into it of water-worn boulders from the surface. But the finding of large rounded stones in a vein at a depth of several hundred feet from the surface is not uncommon, and is explained in this and other instances as resulting from the solvent action of percolating waters in the vein-filling itself.

In connection with the genesis of the ore-bodies of these veins, the hills among which they are found are, in their very origin, the best evidence of the great dynamic movements, evidenced in upheavals, subsidences and foldings which have taken place in the rock-formations during long ages, and have brought about the conditions which favor ore-deposition. (See, as an illustration of this statement, Fig. 3.) The movements which have taken place in this district have been accompanied by numerous and widespread fracturings and lines of weakness, along which the eruptives have made their way and come to the surface. In other words, a region of weakness has come to be a region of fissurings, affording paths for circulating waters. Given the material from which such waters could derive metallic substances, we have the primary conditions necessary to the deposition of ore.

In ore-formation the chemical agencies are many and varied. Gold is soluble in persulphate of iron and in alkaline sulphides, which are present in most underground waters, and gold enters into many unstable combinations with other elements. Once in solution in the underground waters, the gold is distributed by their agency; interchange takes place with the more soluble portions of the rock-mass which these waters permeate; and thus a gradual replacement results in the formation of vein-matter, which gradually accumulates into ore-bodies where the conditions are most favorable.

It has long been widely recognized that iron, zinc, lead, antimony, copper and other metals occurring in metalliferous veins are traceable to the mineral silicates (such as augite, mica and hornblende) common to crystalline rocks; and recent contributions on the subject of the genesis of ore-deposits hold that the heavier metals of any mineral mass have their origin in the magmas of the igneous rocks. Also that the ore-bodies developed in our gold- and silver-mines are the result of repeated concentration of the materials derived from either the igneous magmas or from the general rock-mass, or, where both eruptive and sedimentary rocks exist, from both sources, the deposition being brought about through the agency of underground circulating waters holding the elements of these minerals in solution. The origin of the mineral in the veins of Gilpin county offers in its essentials nothing contradictory to such a theory of genesis.

As the crystalline-metamorphic rocks of this region have an area several hundred times that of the eruptive magmas, to the former (though carrying the elements of ore in smaller proportion than the more basic eruptive material) quite as much as to the latter must be ascribed the origin of the veins of Gilpin county.

A Study of the Elimination of Impurities from Copper-Mattes in the Reverberatory and the Converter.

BY EDWARD KELLER, BALTIMORE COPPER WORKS, BALTIMORE, MD.

(Atlantic City Meeting, February, 1898.)

ABOUT a dozen years ago the art of bessemerizing copper-matte, brought to these shores from France, was first established at the smelter, in Butte, Montana, of the Parrot Silver and Copper Company, which had bought the process from Mr. Manhés, the inventor.

Through the efforts of Mr. J. E. Gaylord, general manager, and Mr. A. J. Schumacher, superintendent, assisted by an able staff, the new method rapidly underwent, at the Parrot establishment, an evolution towards higher perfection, and within a few years its success and its advantages were so marked that all the great works in the west followed in the footsteps of this pioneer.

The converter-process has now practically superseded the reverberatory-process in this country. The latter will undoubtedly, in a not very distant future, be relegated to metallurgical history.

The writer has availed himself of what was perhaps the last chance to make comparisons of the two methods, working practically on the same material on an extensive scale, the reverberatory process being that of the Baltimore Copper Smelting and Rolling Company,* while the converter-process is that of the Anaconda Copper Company—both using the matte from the latter company's works in Anaconda, Montana.

For the greater portion of the material considered in this paper, the writer is indebted to the two companies.

In order to understand the difference of behavior of the several elements, and the difference of the copper produced in the two processes, it is best to study and follow those elements

* This company discontinued the treatment of matte in the spring of 1895.

through the successive stages of metallurgical treatment in the reverberatory-process. The elements that will be considered are lead, bismuth, antimony, arsenic, selenium and tellurium. The latter two will be treated jointly. Zinc will be mentioned in a few instances. Nickel was also determined in the matte, and found to follow the copper, but the quantity in the latter, and the degree of its elimination in the process, was not ascertained.

It should be remarked, however, that the study of the various metallurgical stages, as referred to above, did not always result in positive data. It should, therefore, more properly be called an attempt. The difficulties of the investigation were not chemical. They were encountered in the obtaining of truly corresponding samples and weights of the various materials. These difficulties, as a rule, are insurmountable, for reasons of economy, etc., at most of the smelting-works. Nevertheless, the relative figures obtained and the analyses of the various products may possess some general interest and value.

Reverberatory-Calcining.

The first step in the reverberatory-process, after crushing and rolling the matte to such a fineness that the coarsest particles will pass a 0.2-inch mesh sieve, is the calcining; which, in the case here considered, is accomplished in reverberatory furnaces of the capacity of 15 tons per 24 hours, operated by hand. The analysis of the raw and the calcined matte gives the following result:

Composition Per Cent. of Raw and Calcined Matte.

	Cu.	S.	SO_4.	Fe.	Fe_3O_4.	Zn.	Pb.
Raw, . .	60.89	23.22	......	12.28	0.34	1.70	0.568
	Bi.	Sb.	As.	Se, Te.	Ag.	Au.	
	0.0501	0.1010	0.0481	0.0101	61.1*	0.20*	
	Cu.	S.	SO_4.	Fe.	Fe_3O_4.	Zn.	Pb.
Calcined, .	61.45	16.45	0.99	10.45	3.03	1.66	0.540
	Bi.	Sb.	As.	Se, Te.	Ag.	Au.	
	0.0450	0.0967	0.0370	0.0082	62.2*	0.20*	

These figures show mainly a change in the sulphur-contents. Little sulphate is formed, and only 0.06 per cent. of the copper

* Ounces per ton.

has become soluble in water. About one-fourth of the iron has been converted into magnetic oxide.

It would be assuming too much accuracy in sampling and analysis to deduce from the above figures values for losses of the minor elements.

To obtain some positive information as to quantity of loss of some elements in calcining, a number of experiments were made on a small scale, and the loss of antimony and arsenic was determined. The raw matte was subjected to a dead roast. The contents in the calcined matte were calculated on the original weight of the raw matte. The following are the results per cent:

	Raw Matte.	Calcined Matte.	Difference.	Loss.
Sb, . . .	0.079	0.075	0.004	5.0
As, . . .	0.045	0.038	0.007	15.5

In the partial calcining of actual practice, the loss of the two elements is undoubtedly much smaller.

In order to ascertain the relative degree of elimination of the various elements in the calcining-process, a general sample of the dust from the extensive culverts was analyzed, and found to contain in part:

Partial Analysis of Flue-Dust from Matte-Calcining.

	Cu.	Zn.	Pb.	Bi.	Sb.	As.	Se, Te.
Per cent., . .	18.74	1.40	0.345	0.0205	0.202	0.382	0.014

The simplest manner of solving the problem, above stated, is to bring the figures for the flue-dust and those of the raw matte to a common basis. This is done by calling the copper in each 100, and computing the other elements in proportion. This implies somewhat of error, which, however, does not appreciably affect the relative results. We thus find:

	Cu.	Zn.	Pb.	Bi.	Sb.	As.	Se, Te.
Flue-dust, . . .	100	7.37	1.82	0.110	1.060	2.010	0.070
Raw matte, . . .	100	3.97	0.97	0.069	0.130	0.074	0.025

By dividing the figures for the flue-dust by the figures for the raw matte, the relative elimination in calcining, as indicated by the flue-dust, is found to be:

Relative Elimination of Elements in Matte-Calcining.

Cu.	Zn.	Pb.	Bi.	Sb.	As.	Se, Te.
1.00	1.86	1.88	1.60	8.15	27.16	2.80

It will be observed that the ratio of antimony and arsenic in this case is corroborative of that found in the experiment of dead-roasting.

That portion of volatile elements which escaped through the stacks into the atmosphere is, of course, not taken into consideration; and it is impossible to say what its quantity is, and whether this would be sufficient to alter the relations as found by calculation from the composition of the flue-dust.

The corresponding weights of matte and flue-dust being unknown, it is not possible to calculate quantitatively the elements in the latter. It is, however, possible to arrive at such results approximately in an indirect, more or less theoretical, way. Assuming the loss of arsenic in calcining to be proportional to the elimination of sulphur, we can deduce an approximate figure for the elimination of the elements contained in the flue-dust. In actual calcining, a round one-third of the sulphur is expelled. The elimination of arsenic, as determined for dead-roasting, should then be reduced to one-third. Instead of 15 per cent., we should have an actual loss of 5 per cent. of arsenic.

The ratios of copper and arsenic, as gone into the flue-dust, being 1 : 27, the copper in the flue-dust, in the case under consideration, would thereby be found to be 0.18 per cent. of the copper in the matte. Multiplying this figure for copper by the figures of relative elimination, as given above, we then find the following amounts of the original quantities in the matte as gone into the flue-dust:

Amount of Elements in Matte Gone into Flue-Dust (Theoretical).

	Cu.	Zn.	Pb.	Bi.	Sb.	As	Se,Te.
Per cent., ...	0.18	0.34	0.34	0.29	1.47	5	0.50

Reverberatory Matte-Smelting.

The matte, after being calcined, is subjected to a simple smelting process, accomplished, like the subsequent ones, in a reverberatory furnace. The object of this smelting is the elimination of the iron by slagging, and the production of as pure a subsulphide of copper as possible. A mere melting down of the calcined matte with fluxes, to produce slag, will give the desired result, as will be shown by the following figures:

We have in the calcined matte: Cu, 61.45; S, 16.45; total,

77.90 per cent. Not considering the other elements, and employing the equations: $77.9 : 100 = 61.45 : x$, and $77.9 : 100 = 16.45 : y$, in which x and y stand for the copper and sulphur sought in the resulting product, we find them to be, in round figures: Copper, 79 per cent., and sulphur 21 per cent. This is practically copper-subsulphide, or the "regulus" of the smelter. With the matte as given, this sulphide could only be produced if all the sulphur therein were to combine with the copper, and for that purpose a reducing-flame in the furnace would be necessary to prevent the action of any combined or free oxygen on the sulphur. A neutral or oxidizing flame, which is more generally found in practice, would still produce the same subsulphide, but parallel with that reaction we should have an oxidizing one, eliminating sulphur and forming metallic copper, according to the following chemical equation:

$$Cu_2S + 2O = 2Cu + SO_2;$$

or, as it is found in nearly all metallurgical books:

$$\text{I. } Cu_2S + 3O = Cu_2O + SO_2.$$
$$\text{II. } Cu_2S + 2Cu_2O = 6Cu + SO_2.$$

On this reaction, however it may be expressed, is based the principle of producing metallic copper from copper sulphides.

By this same reaction, in the matte-smelting process, the so-called copper-bottoms are formed along with the sulphide. The less sulphur the calcined matte contains, and the greater the oxidizing influence in the smelting thereof, the greater the amount of these bottoms will be.

In this characteristic the metallurgist has an instrument for producing copper more or less pure, because of the property of the bottoms of carrying a greater amount of impurities than does the remaining sulphide. The greater the quantity of bottoms produced, the more would the supernatant sulphide be drained of impurities, as also of the precious metals, and therefore the purer the copper which could be produced from this sulphide.

A drawback in this process is at once apparent. There is no real elimination of impurities. What one part of the copper gains in quality another part loses.

These bottoms, moreover, have the peculiarity of penetrating the brickwork and furnace-bottoms, by reason of which small 10- to 15-ton furnaces may absorb from 50,000 to 100,000 pounds of copper, in which the silver would be found in double the normal amount, while gold would often be concentrated to ten times its normal quantity.

Our present interest is centered in the behavior of the copper-bottoms (metal) and the regulus (sulphide) towards the impurities. Here, also, data as to quantities of the two produced from a given quantity of matte are absent, and only relative figures can be arrived at.

A somewhat general sample of copper-bottoms derived from the calcined matte, of which the analysis has been given, shows the following contents of impurities:

Impurities in Copper-Bottoms.

	Pb.	Bi.	Sb.	As.	Se,Te.
Per cent., ...	0.398	0.128	0.265	0.343	0.0022

Calling the copper in the calcined matte 100, and computing the impurities in the same proportion, we obtain the amount of impurities (barring a slight error) that would be in the copper if none of those impurities were eliminated.

Impurities that Would be in Copper from Calcined Matte, if None of Them were Eliminated.

	Pb.	Bi.	Sb.	As.	Se, Te.
Per cent., ...	0.879	0.0732	0.1573	0.0602	0.0133

With these latter figures those of the copper-bottoms are directly comparable, and by dividing those of the bottoms by those of the matte we obtain the relative concentration of each element in the copper-bottoms. We have, accordingly:

Concentration of Impurities in Copper-Bottoms.

Cu.	Pb.	Bi.	Sb.	As.	Se, Te.
1	0.45	1.75	1.68	5.70	0.16

As is at once evident, the arsenic has concentrated more than three times as much as antimony or bismuth. Lead, selenium, and tellurium have concentrated in the regulus.

For arsenic, bismuth and antimony this concentration into the copper-bottoms means an evasion of the eliminating influ-

ences of oxidation, and consequent volatilization or slagging, the evasion being necessarily proportional to the degree of concentration.

An analysis of the regulus gave the following results:

Composition of Regulus.

	Cu.	S.	Fe.	Zn.	Pb.	Bi.	Sb.	As.	Se, Te.	Ag.*	Au.*
Per cent.,	81.38	15.90	0.75	0.031	0.216	0.0244	0.0633	0.0411	0.0102	78.5	0.11

Very likely there is also some oxygen present. The sulphur falls far short of the theoretical proportion for the subsulphide.

By comparing the contents of the regulus with those of the calcined matte, as was done with the copper-bottoms, we find the degree of elimination of the elements from the regulus by slagging, volatilization, and draining into the copper-bottoms:

Elimination of Impurities from Regulus.

	Zn.	Pb.	Bi	Sb.	As.	Se, Te.	Ag.	Au.
Per cent., .	98.59	69.85	59.02	50 60	16.14	6.00	4.65	58.46

The product from the matte-smelting furnace, which, undoubtedly, is the main channel for eliminating impurities, is the slag.

Its source is the iron of the matte and the siliceous materials, with a little lime, added as flux. A sample of such a slag was found to be composed as follows:

Analysis of Matte-Smelting Slag.

	SiO_2.	FeO.	Fe_3O_4.	Al_2O_3.	CaO.	ZnO.	Cu.	Pb.	Bi.	Sb.	As.	S.
Per cent., .	29.68	22.99	18.00	9.62	3.75	3.80	10.88	0.80	0.008	0.071	0.014	0.341

This slag is remarkable for the fact that the copper therein is in a metallic state, and for its high percentage of magnetic oxide. Its magnetic properties can be judged from the further fact that a one-pound horse-shoe magnet will lift pieces of from three to four grammes.

Selenium and tellurium are present in this slag in but very small quantity. They were not determined.

If, as with the other products heretofore described, we bring slag and matte to the basis of copper = 100, and divide the fig-

* Ounces per ton.

ures for the slag by those for the matte, we obtain the relative slaggability of the elements in matte-smelting. The figures are:

	Cu.	Zn.	Pb.	Bi.	Sb.	As.
Slag, ...	100	28.03	7.350	0.0730	0.6520	0.1290
Matte, ...	100	2.70	0.879	0.0732	0.1573	0.0602

Relative Slaggability in Matte-Smelting.

Cu.	Zn.	Pb.	Bi.	Sb.	As.
1	10.38	8.36	1	4.15	2.14

Although knowledge of the quantity of slag, by weight, from a given quantity of matte is lacking, it is nevertheless possible to calculate the absolute amount of such slag to its corresponding amount of matte; and thereby also the actual amounts of impurities eliminated. We know the amount of iron in the calcined matte to be 12.5 per cent. We also know that practically all of this has gone into the slag (the latter receiving iron from no other source), of which it constitutes 36.5 per cent. These are sufficient data to make the calculation by a simple equation. We find for 100 units of matte in round figures 41 units of slag; or for 100 of metallic copper 67 of slag. Again, by another equation we find the impurities in 41 units of slag, or the amounts eliminated from 100 units of matte. The two compare as follows:

Comparative Contents in Matte and Slag, Per Cent.

	Cu.	Zn.	Pb.	Bi.	Sb.	As.
Matte, ...	61.45	1.660	0.540	0.0450	0.0967	0.0370
Slag, ...	4.461	1.251	0.328	0.0033	0.0291	0.0057

From these two series of figures the percentage of elements slagged in this special slag is found to be:

Elements Slagged in Matte-Smelting.

	Cu.	Zn.	Pb.	Bi.	Sb.	As.
Per cent., . . .	7.26	75.36	60.7	7.3	30.1	15.4

The above-described slag is abnormally high in copper, and, therefore, the impurities are very likely correspondingly above the normal.

The following is the result of a partial analysis of a slag of lower grade:

Composition of Matte-Smelting Slag (Partial Analysis).

	Cu.	Fe.	Pb.	Bi.	Sb.	As.
Per cent., . . .	3.34	37.97	0.3639	0.0052	0.0184	0.0050

Proceeding in the same manner as explained for the first slag, we find the elimination of impurities in this slag to be:

Elements Slagged in Matte-Smelting.

	Cu.	Pb.	Bi.	Sb.	As.
Per cent., ...	1.1	22.2	3.8	6.2	4.3

The average amount of slagging, as judged from the average amount of copper found in such slags, is undoubtedly close to the mean of the two extremes given, or:

Average Amount of Elements Slagged in Matte-Smelting.

	Cu.	Pb.	Bi.	Sb.	As.
Per cent., ...	4.18	41.5	5.6	18.2	9.9

The matte-smelting process also produces its flue-dust; but as the heat in this process is very high and the chambers are not very extensive, it is quite certain that much of the volatile products escape into the atmosphere, and that the elements as found in the flue-dust do not as truly represent the elimination through this channel as they do in the calcining process.

A partial analysis of the matte-smelting flue-dust showed the following results:

Composition of Flue-Dust from Matte-Smelting (Partial Analysis).

	Cu.	Zn.	Pb.	Bi.	Sb.	As.	Se, Te.
Per cent., .	36.40	0.622	0.039	0.062	0.0713	0.0631	0.0128

The relative amounts of the elements in this flue-dust were calculated as follows:

Relative Amounts of Elements Gone Into Flue-Dust in the Matte-Smelting.

Cu.	Zn.	Pb.	Bi.	Sb.	As.	Se, Te.
1	0.64	0.13	2.32	1.27	2.99	2.70

PRODUCTION OF BLISTER-COPPER FROM REGULUS.

In the reverberatory-process which we are following, but one grade of copper is produced. Copper-bottoms and regulus are treated together in the blister-furnace. In this furnace the

whole operation consists in the reaction already described, namely:

$$\text{I. } Cu_2S + 3O = Cu_2O + SO_2.$$
$$\text{II. } Cu_2S + 2\,Cu_2O = 6\,Cu + SO_2.$$

This is analogous to the formation of copper-bottoms, with which we became acquainted in matte-smelting, and in which we found the elements concentrated in the following order: arsenic, bismuth, antimony, lead, selenium and tellurium; their degree of escape from the eliminating influence being in the same order.

There being no appreciable quantity of iron in the regulus, the formation of slag must, in this smelting, be necessarily much lighter than in that of the matte.

An analysis of such a slag showed the following contents of copper and impurities:

Composition of Blister-Slag (Partial Analysis).

	Cu.	Pb.	Bi.	Sb.	As.	Se, Te.
Per cent., . .	20.43	0.1054	0.0049	0.0320	0.0039	0.0025

Comparing these figures with those of the regulus, the relative slaggability in the blister-process is found to be:

Relative Slaggability of the Elements in the Blister-Process.

Cu.	Pb.	Bi.	Sb.	As.	Se, Te.
1	1.95	0.80	2.02	0.38	0.98

Quantities cannot be given in this case.

Owing to the solid character of the material smelted, the production of flue-dust is necessarily light; the volatilization of some of the elements is, by reason of the comparatively high heat, probably great.

The sample taken (a block of 9 by 10 by 5 inches) of the blister-copper produced in this process, proved too unsatisfactory (due to the concentration of the impurities towards the center) to allow of deductions being made as to the elimination of those impurities.

For completeness' sake, analyses of three samples from the block of blister-copper are herewith given:

Impurities in Block of Blister-Copper.

	Top. Per cent.	Center. Per cent.	Bottom. Per cent.
Pb,	0.019	0.058	0.002
Bi,	0.024	0.055	0.005
Sb,	0.099	0.157	0.048
As,	0.074	0.108	0.034
Se, Te,	0.019	0.027	0.004
S,	0.047	0.112	0.040
Ag,	0.454	0.641	0.248
Au,	0.00089	0.00109	0.00068

Refining of Blister-Copper.

All blister-copper at the works mentioned is subjected to a furnace refining-process, and after this it is cast into anodes for the multiple electrolytic system, or into cakes, which must undergo the serious test of rolling, to form anodes for the series electrolytic system.

This refining-process consists in melting down the pigs of blister-copper in an oxidizing atmosphere and beating the surface of the molten metal with a rabble (an operation called flapping), thus increasing the surface, and facilitating the oxidation of the impurities. The elimination of all the sulphur is essential. When all of this is expelled, a part of the copper has also become oxidized; the suboxide being to a great extent soluble in the molten metal. The slag is carefully skimmed, and, to reduce the copper suboxide in the molten copper, the process of poling is performed. This latter operation consists in forcing large poles of green wood into the full depth of the molten furnace-charge. Hydrocarbon gases are thus evolved which reduce the oxides. If any slag be left on the surface, the impurities therein will naturally be re-precipitated into the copper, deteriorating its quality.

The oxygen is never completely removed by the poling. With a certain admixture of suboxide the copper is said to possess the best "set"—that is, on solidification the surface shows neither a rising nor a sinking, at which point the physical properties are claimed to show most favorably.

Having no accurate sample, and therefore no absolutely reliable data as to the contents of impurities in the blister-copper, it is impossible to show the positive amount of elimination in this important process.

From a slag which was analyzed we are able to calculate its contents of impurities back to what amount they represented in the blister-copper, and we can compare these with the quantities remaining in the refined copper.

The results of the slag-analysis are as follows:

Composition of Refinery-Slag (Partial Analysis).

	Cu.	Pb.	Bi.	Sb.	As.
Per cent., . .	27.79	0.9694	0.0129	0.1640	0.0289

In this slag we know the copper to be, in round numbers, 2 per cent. of the blister-copper from which it is derived, and on this basis find the slag to be 7 units to every 100 of copper. By a simple equation we find the impurities in the slag, expressed in percentage as contained in the blister-copper, to be as follows:

Impurities in Blister-Copper Slagged.

	Per cent.
Pb, ...	0.0678
Bi, ...	0.0009
Sb, ...	0.0115
As, ...	0.0020

Impurities in Refined Copper.

	Per cent.
Pb, ...	0.0093
Bi, ...	0.0320
Sb, ...	0.0651
As, ...	0.0586
Se, Te, ...	0.0098

In the following the results are derived from converter blister-copper. In the latter the true contents of impurities are also unknown, for the same reasons of difficulties in sampling, as have been pointed out before. The tests were therefore of necessity confined to the refinery-products—copper, slag, and the dust that could be collected. The quantities of the three products were: refined copper, 2,450,000 pounds; refinery-slag, 63,387 pounds; dust, 290 pounds. They gave the following analytical results:

Composition of Refined Converter-Copper.

	Cu.	Ag.	Pb.	Bi.	Sb.	As.	Se, Te.	O.
Per cent.,	99.25	0.36 (=105 ozs.)	0.0103	0.0040	0.0630	0.0211	0.0072	0.284

The slag consisted of 10 parts slag proper to 1 part of metallic scales:

Composition of Refinery-Slag.

Metallic Scales.

	Per cent.
Cu,	98.95
Ag,	0.375 (= 109.4 ozs.)
Pb,	0.0293
Bi,	0.0044
Sb,	0.0680
As,	0.0490
Se, Te,	0.0101

Slag Proper.

	Per cent.
SiO_2,	39.02
Cu,	35.66
Fe,	8.21
Al_2O_3,	4.19
CaO,	5.04
Pb,	0.65
Bi,	0.0018
Sb,	0.2180
As,	0.0490
Se, Te,	0.0018
S,	0.61
O,	6.01
Ag,	0.05 (= 14.5 ozs.)

Average of Slag Proper and Metallics.

	Per cent.
Cu,	44.47
Pb,	0.5936
Bi,	0.0020
Sb,	0.2044
As,	0.0490
Se, Te,	0.0026

The refinery-dust consists merely of the sweepings from the roofs around the furnace-stacks:

Composition of Refinery-Dust.

	Cu.	Pb.	Bi.	Sb.	As.	Se, Te.
Per cent.,	92.60	0.004	0.0114	0.065	0.047	0.0064

The quantities in this dust are too small to be considered. The relative quantities of the elements gone into it are, however, of some interest. They were calculated to be:

Relative Amounts of Elements Gone into Refinery-Dust.

Cu.	Pb.	Bi.	Sb.	As.	Se, Te.
1	1.06	3.04	1.03	2.28	0.95

Some of the pieces of metallic copper in the slag were tested for oxygen, and as much as 1.59 per cent. of that element, or 14.19 per cent. of copper suboxide, was found.

The silver is seen to be concentrated from the normal of 105 ounces in the refined copper to 109.5 ounces in the metallics in the slag. In the slag proper the silver of 14.5 ounces corresponds to 40 ounces per ton of copper therein.

As we do not know the original amount of impurities in the copper before refining, it seems simplest to compare the amounts slagged with those in the refined copper. In this way it is possible to make comparisons of the refining results of coppers of different grades.

In the example just stated the actual amount of copper slagged is 1.15 per cent. of the refined copper. Reducing this to 1, and the percentages of the impurities in the same proportion, we have the relative slaggability in this special case:

Relative Slaggability in Refining Converter-Copper.

Cu.	Pb.	Bi.	Sb.	As.	Se, Te.
1	129.50	1.13	7.30	5.22	0.81

The following are the results from an electrolytic copper, and the slag from its refining: copper, 50,000 pounds; slag, 670 pounds.*

Impurities in Electrolytic Copper, after Furnace-Refining.

	Pb.	Sb.	As.
Per cent., ...	0.00093	0.00164	0.00028

Copper and Impurities in Slag.

	Cu.	Pb.	Sb.	As.
Per cent., ...	45	0.0173	0.0073	0.00089

Copper and Impurities in Slag Corresponding to Percentage in Copper-Charge.

	Cu.	Pb.	Sb.	As.
Per cent., ...	0.6	0.00023	0.000098	0.000012

Relative Slaggability in Refining Electrolytic Copper.

Cu.	Pb.	Sb.	As.
1	41	9.97	7.13

In comparing various coppers and slags in this manner it is found that the figures for bismuth, antimony, and arsenic are almost constant, though the quantities in one copper may be many hundreds of times greater than those in another. Lead

* The sampling of electrolytic copper, like that of bar-copper, is only satisfactory after the cathode-plates are melted down and the furnace-charge has become thoroughly mixed; because not only different plates from the same tank, but different portions of the same plate, show different composition.

seems more variable, the relative slaggability decreasing as its quantity in the copper decreases.

The Reduction of the Slags from the Reverberatory Process.

Where matte is smelted by the reverberatory process the slags from the refining-furnaces, generally containing about 40 per cent. of copper in the form of suboxide, are returned to the blister-furnace, there at once aiding in the production of metallic copper from the regulus or subsulphide of copper.

To regain the copper from the slags of the blister- and matte-smelting furnaces, these must be reduced in a cupola or blast-furnace with the aid of iron pyrites, whence will result a secondary matte, and a slag clean enough to be discarded.

The following is an example of a smelting-mixture of such material:

	Pounds.
Coke,	350
Matte-smelting slag,	2500
Blister-slag,	300
Pyrites,	600
Carbonate of lime,	400

It should be remarked that of the sulphur in the pyrites, there is never more than one atom metallurgically utilized in the formation of matte, the second atom being consumed as fuel, and thus only its calorific value being obtained.

To 100,000 pounds of refined copper about 5300 pounds, or 5.3 per cent., of copper would return in the form of the secondary matte, to go through the whole process over again.

The following analyses pertain to the secondary products:

Composition of Secondary Matte.

	Per cent
Cu,	43.13
S,	23.10
Fe,	26.18
Fe_3O_4,	0.61
Zn,	2.07
Ag, Au,	0.0840
Pb,	1.4260
Bi,	0.0071
Sb,	0.2677
As,	0.0437
Se, Te,	0.0050
SiO_2,	2.97

Secondary Matte, Calcined.

	Per cent.
Cu,	42.28
S,	17.46
SO_4,	0.91
Fe_3O_4,	11.60

This matte is more difficult to calcine than the primary, or original, matte, the calcined product of the former always showing more sulphur, when subjected to the same amount of handling in the calciners, than the latter. About one-third of the iron in the raw matte is converted into magnetic oxide. It also differs from the original matte in that it requires two smeltings to be converted into regulus, the first smelting yielding a product of 65 to 70 per cent. copper, called white metal by the smelter.

Through this extra operation some of the impurities are eliminated to a greater extent than in the regular course.

The following figures show the impurities in the refined copper derived from secondary matte, and the amount of impurities that would be in the copper if none were eliminated, from which figures the elimination is calculated:

Impurities in Matte; Cu = 100.

	Per cent.
Pb,	3.3063
Bi,	0.0165
Sb,	0.6207
As,	0.1013
Se, Te,	0.0116

Impurities in Copper.

	Per cent.
Pb,	0.0208
Bi,	0.0109
Sb,	0.0800
As,	0.0275
Se, Te,	0.0067

Elimination.

	Per cent.
Pb,	99
Bi,	34
Sb,	87
As,	73
Se, Te,	42

Knowing that the copper in the secondary matte is 5.8 per cent. of the total copper, we are enabled to calculate from the

above figures, and the contents of the original matte, the percentage of original impurities entering the metallurgical circuit a second time, and we find in round numbers:

Copper and Impurities from Original Matte Contained in Secondary Matte.

	Cu.	Pb.	Bi.	Sb.	As.	Se, Te.
Per cent., . .	5.3	18.	1.	25.	7	2.5

The following is an analysis of a representative sample of the blast-furnace slag from slag-reduction:

Composition of Blast-Furnace Slag.

	SiO_2.	FeO.	Al_2O_3.	CaO.	ZnO.	Cu.	Pb.	Sb.	As.
Per cent., .	36.55	43.97	6.56	10.05	2.01	0.27	0.26	0.0317	0.0033

It will be noticed that the elimination of lead, antimony and arsenic through this channel is very appreciable.

The Reduction of Refinery-Slags in the Blast-Furnace.

At works where metallic copper alone is treated, the refining-slags must also be reduced by means of a blast-furnace. A full analysis of such a slag has been shown (page 139). The following may be given as one of the smelting-mixtures:

	Pounds.
Refinery-slag, ...	500
Coke, ...	300
Carbonate of lime, ...	500
Dump-slag (high in iron), ...	100
Cobbings (material from furnace-repairing), ...	100
Refinery-slag, ...	500
Coke, ...	500

The resulting products from this smelting are a black-copper, some matte, and a clean slag which is discarded. Subjoined are some analyses of the copper and slag:

Impurities in Black-Copper from Refinery-Slags.

	I. Per cent.	II. Per cent.
Fe, ...	0.17	
S, ...	0.796	
Pb, ...	0.780	0.3656
Bi, ...	0.0035	0.0041
Sb, ...	0.2380	0.2424
As, ...	0.0520	0.1852
Se, Te, ...	0.0095	0.0122
Ag, ...	62.2 oz. per ton.	
Au, ...	0.14 " " "	

Composition of Blast-Furnace Slag (from Reduction of Refinery-Slag).

	SiO_2.	FeO.	Al_2O_3.	CaO.	Cu.	Pb.	Sb.	As.
Per cent., .	44.43	6.43	4.22	43.40	0.69	0.3064	0.0174	0.0059

The final loss of copper through this slag is figured to be from 0.03 to 0.04 per cent. of the total copper refined.

The antimony eliminated through this channel is 2.4 per cent. of that in the refined copper, and the arsenic is 0.21 per cent., expressed in the same way.

Analysis II., given above, is of a sample from 642,000 pounds of black-copper which itself corresponded to 33,500,000 pounds of refined copper; the former is, therefore, 1.9 per cent. of the latter.

The impurities in the refined copper were found to be:

Impurities in Refined Copper.

	Pb.	Bi.	Sb.	As.	Se, Te.
Per cent., . . .	0.0082	0.0035	0.0293	0.0416	0.0076

We are thus able to calculate what percentage the impurities in this black-copper constitute of the impurities in the corresponding refined copper.

Impurities in Black-Copper as Figured to Refined Copper.

	Pb.	Bi.	Sb.	As.	Se, Te.
Per cent., ...	85.5	2.22	15.71	8.45	3.04

These figures are for 1.9 per cent. of copper actually slagged. Reducing them to copper as a unit, we find them corroborative of similar figures shown before.

Relative Amounts of Copper and Impurities in Black-Copper, Derived from Refinery-Slags.

Cu.	Pb.	Bi.	Sb.	As.	Se, Te.
1	45	1.17	8.27	4.45	1.60

Elimination of Impurities in the Total Reverberatory Process of Copper-Smelting.

Since in the foregoing the results were largely of a qualitative nature, and incomplete on account of the loss of volatile products, it was necessary to pursue a different course to determine the total elimination of the impurities in the whole reverberatory-process.

For that purpose an accurate sample of matte of an eight months' run was taken, and for the corresponding period every month's sample of the resulting refined copper was analyzed. The two products show the following composition:

Composition of Matte.

	Per cent.	
Cu,	60.76	
S,	23.25	
Fe,	11.43	
Fe_3O_4,	1.13	
Zn,	2.41	
Ni,	0.001	
Pb,	0.590	
Bi,	0.042	
Sb,	0.079	
As,	0.045	
Se, Te,	0.015	
Ag,	60.4	ozs. per ton.
Au,	0.30	oz. "

Impurities in Refined Copper.

	Per cent.
Pb,	0.0093
Bi,	0.0320
Sb,	0.0651
As,	0.0586
Se, Te,	0.0098

From the above it is easy to calculate the total elimination of impurities in the reverberatory-process. The round figures were found to be:

Elimination of Impurities in the Reverberatory-Process.

	Per cent.
Pb,	99
Bi,	54
Sb,	50
As,	21
Se, Te,	60

In comparing these results with what was observed in the study of the several stages of the process, we understand without difficulty the relative behavior of arsenic, antimony and lead. Although in calcining, arsenic is many times more volatile than the other elements, the total eliminated in that process is not very large. In the subsequent treatment the arsenic evades elimination to a much greater degree than the others,

by retreating to the so-called bottoms. It is this property that retains arsenic in the copper more than any of the other elements. The elimination of antimony and lead is explained by their affinity to oxygen and the slaggability of their oxides, as also by their volatility.

The comparatively high degree of elimination of bismuth, and of selenium and tellurium, was not positively indicated by the analysis of any of the smelting-products. The relative amount of the first found in the dust from the refining-furnaces was, however, greater than that of the other elements. We are, therefore, forced to infer that these elements were oxidized, and volatilized beyond the dust-chambers. The study of the converter-products will support this inference.

Elimination of Impurities in the Converter-Process.

In the converter-process the several stages of the reverberatory-process, requiring three or four days' operation, are condensed into about one hour's time.

The refined converter-copper, from practically the same matte and the same period of time as the reverberatory-copper above described, was analyzed, and the elimination of impurities was calculated:

Impurities in Matte.

(Cu, 60.76 per cent.)

	Per cent.
Pb,	0.590
Bi,	0.042
Sb,	0.079
As,	0.045
Se, Te,	0.015

Impurities in Converter-Copper.

(Stalmann Converter.)

	Per cent.
Pb,	0.0082
Bi,	0.0025
Sb,	0.0443
As,	0.0068
Se, Te,	0.0071

Elimination.

	Per cent.
Pb,	99
Bi,	96
Sb,	66
As,	91
Se, Te,	71

On about two months' run, the new and much larger Anaconda converter shows the following results:

Impurities in Matte.

(Cu, 60.89 per cent.)

	Per cent.
Pb,	0.5680
Bi,	0.0501
Sb,	0.1010
As,	0.0481
Se, Te,	0.0101

Impurities in Converter-Copper.

(New Anaconda Converter.)

	Per cent.
Pb,	0.0103
Bi,	0.0040
Sb,	0.0630
As,	0.0211
Se, Te,	0.0072

Elimination.

	Per cent.
Pb,	99
Bi,	95
Sb,	62
As,	73
Se, Te,	57

The writer is indebted to Mr. Frank Klepetko for a very interesting collection of samples from the Boston and Montana Co.'s works at Great Falls, Montana. The analytical results and deductions from these samples are subjoined:

Impurities in Blast-Furnace Matte.

(Cu, 61.42 per cent.)

	Per cent.
Pb,	0.0370
Bi,	0.0049
Sb,	0.1330
As,	0.1280
Se, Te,	0.0042

Impurities in Converter-Copper.

(B. & M. Converter.)

	Per cent.
Pb,	0.0040
Bi,	0.0023
Sb,	0.0903
As,	0.0448
Se, Te,	0.0967

Elimination.

	Per cent.
Pb,	94
Bi,	71
Sb,	58
As,	78
Se, Te,	2

Impurities in Reverberatory Matte (I.).

(Cu, 56.32 per cent.).

	Per cent.
Pb,	0.0665
Bi,	0.0267
Sb,	0.0824
As,	0.0282
Se, Te,	0.0038

Impurities in Converter-Copper.

(B. & M. Converter.)

	Per cent.
Pb,	0.0081
Bi,	0.0019
Sb,	0.0435
As,	0.0203
Se, Te,	0.0060

Elimination.

	Per cent.
Pb,	93
Bi,	96
Sb,	70
As,	59
Se, Te,	10

Impurities in Reverberatory Matte (II.).

(Cu, 49.34; Fe, 22.44 per cent.)

	Per cent.
Pb,	0.0738
Bi,	0.0337
Sb,	0.1010
As,	0.0480
Se, Te,	0.0021

Impurities in Converter-Copper.

(Under-blown Charge.)

	Per cent.
Pb,	0.0087
Bi,	0.0056
Sb,	0.0992
As,	0.0260
Se, Te,	0.0020

Elimination.

	Per cent.
Pb,	94
Bi,	92
Sb,	52
As,	73
Se, Te,	52

Matte as Above: Impurities in Regulus (from Under-blown Charge).

(Cu, 81.40 per cent.)

	Per cent.
Pb,	0.0071
Bi,	0.0012
Sb,	0.0365
As,	0.0081
Se, Te,	0.0084

Elimination.

	Per cent.
Pb,	94
Bi,	99
Sb,	78
As,	90
Se, Te,	2.45 concentration.

Matte as Above: Impurities in Converter-Copper (Regularly-Blown Charge).

	Per cent.
Pb,	0.0069
Bi,	0.0029
Sb,	0.0546
As,	0.0156
Se, Te,	0.0034

Elimination.

	Per cent.
Pb,	95
Bi,	96
Sb,	73
As,	84
Se, Te,	19

Matte as Above: Impurities in Converter-Copper (Over-blown Charge).

	Per cent.
Pb,	0.0016
Bi,	0.0015
Sb,	0.0195
As,	0.0072
Se, Te,	0.0033

Elimination.

	Per cent.
Pb,	99
Bi,	99
Sb,	90
As,	93
Se, Te,	21

Comparing the elimination of the elements in the under-blown charge with the elimination in the corresponding regulus, we observe a striking parallel between this partial operation of the converter and the matte-smelting in the reverberatory-process. We see that lead is eliminated in the same degree from metal and sulphide. It does not concentrate in either copper or sulphide.

Bismuth, arsenic and antimony show a greater degree of elimination in the sulphide than in the copper, this being due to their concentration in the copper.

The selenium and tellurium show elimination from the copper, while in the sulphide they show positive concentration; that is to say, the sulphide contains more of these two elements than is contained in the corresponding amount of matte.

If in the copper of the over-blown charge we calculate the elimination based on the impurities in the copper of the regularly-blown charge, we obtain the elimination by over-blowing. The figures are found to be as follows:

ELIMINATION OF IMPURITIES BY OVER-BLOWING REGULAR CHARGE.

Impurities in Regular Copper.

	Per cent.
Pb,	0.0069
Bi,	0.0029
Sb,	0.0546
As,	0.0156
Se, Te,	0.0034

Impurities in Over-blown Copper.

	Per cent.
Pb,	0.0016
Bi,	0.0015
Sb,	0.0195
As,	0.0072
Se, Te,	0.0033

Elimination.

	Per cent.
Pb,	77
Bi,	48
Sb,	64
As,	54
Se, Te,	3

It is evident that a few minutes' over-blowing of a charge of copper in a converter is a more efficient means of eliminating impurities than hours of "flapping" in the reverberatory-process.

The question has often occurred to the writer: Could not the process of furnace-refining be successfully and advantageously completed in the converter by substituting for the pole of green wood of the reverberatory process a blast of pure reducing-gas?

The converter-slag corresponding to the Boston and Montana reverberatory-matte (II.) was analyzed, yielding the following results:

Composition of Converter-Slag (B. & M.).

	Per cent.
SiO_2,	30.26
FeO,	59.56
Al_2O_3,	5.71
CaO,	0.86
ZnO,	0.59
MnO,	trace
Cu,	1.31
Pb,	0.0101
Bi,	0.0008
Sb,	0.0474
As,	0.0105
Se, Te,	0.0

Knowing the amount of iron in the matte and in the slag, and taking it for granted that all of that element has entered the latter, we are enabled, as shown before—weights being unknown—to calculate the amount of slag for a given amount of matte or copper. The figures are found to be: 48 slag to 100 matte; 98 slag to 100 copper.

Now, knowing the amount of original impurities from the matte remaining in the copper, and being able to calculate those gone into the slag, we find by difference the amount of

those impurities volatilized. The results of the calculations are as follows:

Distribution of Original Impurities Contained in Matte.

Impurities in Copper.

	Per cent.
Pb,	5
Bi,	4
Sb,	27
As,	16
Se, Te,	81

Impurities in Slag.

	Per cent.
Pb,	7
Bi,	1
Sb,	23
As,	11
Se, Te,	0

Impurities Volatilized.

	Per cent.
Pb,	88
Bi,	95
Sb,	50
As,	73
Se, Te,	19

These results, of course, pertain to the one special case.

Through the kindness of Mr. D. E. Heller, the writer obtained also a collection of samples from the converter-plant of the Montana Ore Purchasing Co., of Butte City, Montana. The converters of this company are of the Parrot type, and are smaller than those of the other companies. The analytical results of the products are given below:

Impurities in Matte. (Cu, 52.00 per cent.; Fe, 20.88 per cent.)

(Montana Ore Purchasing Co.'s Converter.)

	Per cent.
Pb,	1.2523
Bi,	0.0418
Sb,	0.0950
As,	0.0634
Se, Te,	0.0085

Impurities in Converter-Copper.

(Montana Ore Purchasing Co.'s Converter.)

	Per cent.
Pb,	0.0517
Bi,	0.0051
Sb,	0.0533
As,	0.0231
Se, Te,	0.0078

Elimination.

(Montana Ore Purchasing Co.'s Converter.)

	Per cent.
Pb,	98
Bi,	94
Sb,	71
As,	81
Se, Te,	52

Composition of Converter-Slag.

	Per cent.
SiO_2,	29.97
FeO,	58.40
Al_2O_3,	3.55
CaO,	2.12
ZnO,	2.36
Cu,	1.09
Pb,	1.245
Bi,	0.001
Sb,	0.087
As,	0.021
Se, Te,	0.0
S,	0.250

This slag is said to be a somewhat general sample, and does not correspond to the matte- and copper-sample. If, however, we make the same calculations based on these, as were made with the B. & M. products, we obtain similar results, excepting with lead, of which in this case more seems to be slagged, and less volatilized. The results are as follows: 46 slag to 100 matte; 88 slag to 100 copper.

Distribution of Original Impurities Contained in Matte.

(Montana Ore Purchasing Co.'s Converter.)

Impurities in Copper.

	Per cent.
Pb,	2
Bi,	6
Sb,	29
As,	19
Se, Te,	48

Impurities in Slag.

	Per cent.
Pb,	46
Bi,	1
Sb,	42
As,	15
Se, Te,	0

Impurities Volatilized.

	Per cent.
Pb,	52
Bi,	93
Sb,	29
As,	66
Se, Te,	52

Composition of Converter Flue-Dust.

(Montana Ore Purchasing Co.'s Converter.)

Chambers Near Converter.

	Per cent.
Cu,	23.32
Fe,	3.30
Zn,	2.85
Pb,	18.81
Bi,	0.42
Sb,	0.935
As,	1.805
Se, Te,	0.0026
S,	1.34
SO_4,	34.46
H_2O,	10.87
Ag,	18.2 oz.

Chambers Distant From Converter.

	Per cent.
Cu,	2.36
Fe,	2.80
Zn,	5.53
Pb,	32.24
Bi,	0.480
Sb,	2.186
As,	5.694
Se, Te,	0.0054
S,	0.0
SO_4,	36.15
H_2O,	——
Ag,	6.4 oz.

As the matte, converter-copper, and slag have furnished positive data for calculations and theoretical discussion, nothing need be said in that way about the flue-dust.

Others have pointed out the volatility of silver in converting. This is confirmed by the above analyses. Though the copper be considered as mechanically carried over, the silver must be volatile, because in the distant chambers there is much more silver, compared to copper, than there is in the near chambers.

The flue-dusts from the converters, as well as those of the reverberatory-process, were tested as to the solubility of their copper-contents. The results of these tests are given in the table below:

Solubility of Copper Contained in Flue-Dust.

Flue-Dust.	Total Copper.	Copper Soluble in Water.	Of the Remaining Copper Soluble in Hunt & Douglas's Solution.	Copper Insoluble in the Two Reagents.
Matte-calcining......	18.74 per cent.	17.66 per cent.	....	....
Matte-smelting......	36.40 "	12.26 "	18.74 per cent.	5.40 per cent.
Converter, near chambers...	23.32 "	11.63 "	8.37 "	3.32 "
Converter, distant chambers...........	2.36 "	2.31 "	....	....

These results go to show how, through defective dust-chambers (surface or under-ground), by the access of rain-, ground-, or tide-water, severe losses in copper may accrue. They further suggest the possibility of extracting comparatively very pure copper from a very impure material; eliminating the noxious elements from circulation in the copper-plant, and turning them over, as a more fit material, to the lead-smelter.

Of the results obtained from the several converters, only those are comparable, as to the merits of eliminating impurities, in cases where the matte and the resulting copper are treated in the same manner. The mattes should, further, contain about the same amount of impurities.

The Stalmann and the new Anaconda converter are directly comparable. The matte for both is nearly the same, and before entering either, it was remelted in a cupola-furnace. The copper from both is refined copper. With the copper from these two converters may also be compared that from the reverberatory-process; it also being refined copper.

With the B. & M. Co.'s converter is comparable that of the M. O. P. Co.; taking, for the former, reverberatory-matte (II.).

To these converters the matte passes directly from the matting-furnaces. The copper is crude or blister-copper.

In the table on the following page are given the capacities of the several converters, the results of their working, and that of the reverberatory process.

Lead, bismuth, antimony and arsenic show comparatively small variations in the several converters. Selenium and tellurium seem to be decidedly less eliminated in the larger than in the smaller ones.

To determine the order of degree of elimination of the several elements we cannot take one single matte or converter. Were all the elements present to the same amount in any one matte, the percentage of their elimination would be the true value of this metallurgical relation, and would enable us to place them in a proper series.

Varying amounts, however, change the relative position of the elements as regards degree of elimination. This is illustrated by the behavior of lead and bismuth. In every instance but one, the lead-contents in the matte exceed many times those of bismuth, and when this is the case lead shows a greater degree of elimination than bismuth. Where the quantities of the two approach equality, as in B. & M. reverberatory-matte (II.), bismuth stands first. The same is noticed with arsenic and antimony, the former element being, in general, much more readily eliminated. But in B. & M. reverberatory-matte (I.) the quantity of antimony is three times that of arsenic, and the order of degree of elimination is reversed.

For properly fixing the positions of the elements in the elimination-series the B. & M. mattes and converters offer a favorable opportunity.

In the blast-furnace matte the antimony and arsenic are contained in approximately the same quantities: Sb, 0.133; As, 0.128 per cent. Their elimination is: Sb, 58; As, 78 per cent. These two figures may be taken as the true relative value of the two elements as regards their elimination by converting in this special converter and for the special tenors named. For each of the following pairs the same proposition holds true. In the same matte, bismuth is comparable with selenium and tellurium, the tenors being: Bi, 0.0049; Se, Te, 0.0042 per cent; and the elimination: Bi, 71; Se, Te, 2 per cent.

Comparison of Different Converters.

Process	Element.	Reverberatory-Process.	Stalmann Converter.	New Anaconda Converter.	B. & M. Co. Converter.	M. O. P. Co. Converter.
Capacity			Initial Charge, 3,000 Lbs. / Maximum Charge, 9,000 Lbs.	Initial Charge, 7,000 Lbs. / Maximum Charge, 17,000 Lbs.	Initial Charge, 10,000 Lbs. / Maximum Charge, 22,000 Lbs.	Initial Charge, 2,500 Lbs. / Maximum Charge, 9,000 Lbs.
Matte.		Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
	Pb.	0.590	0.590	0.5680	0.0738	1.2523
	Bi.	0.042	0.042	0.0501	0.0337	0.0418
	Sb.	0.079	0.079	0.1010	0.1010	0.0950
	As.	0.045	0.045	0.0481	0.0480	0.0634
	Se, Te.	0.015	0.015	0.0101	0.0021	0.0085
Copper.	Pb.	0.0093	0.0082	0.0103	0.0069	0.0517
	Bi.	0.0320	0.0025	0.0040	0.0029	0.0051
	Sb.	0.0651	0.0443	0.0630	0.0546	0.0533
	As.	0.0586	0.0068	0.0211	0.0156	0.0231
	Se, Te.	0.0098	0.0071	0.0072	0.0034	0.0078
Elimination.	Pb.	99	99	99	95	98
	Bi.	54	96	95	96	94
	Sb.	50	66	62	73	71
	As.	21	91	73	84	81
	Se, Te.	60	71	57	19	52
Remarks			Matte remelted in cupola.		Matte from matting-furnace directly to converter.	
		Refined copper.			Blister copper.	

In reverberatory-matte (II.) the contents of bismuth and lead approximate more closely than in any of the others, Bi being 0.0337 and Pb, 0.0738 per cent. Here the elimination is: Bi, 96; Pb, 95 per cent. From these figures it follows that the elements named stand as follows:

Elimination-Series of Elements for B. & M. Converter.

Bi > Pb > As > Sb > Se, Te.

For the reverberatory-process it is evident that the following series holds good:

Elimination-Series of Elements in the Reverberatory-Process.

Pb > Se, Te > Bi > Sb > As.

If we wish to explain the differences of behavior of the elements in the two processes we must recall the results obtained by analysis of the various products. Bismuth, it was seen, is the most volatile of all in the converter-process, little of it remaining in the copper, and still less being retained by the slag. In the reverberatory-process it greatly evades elimination by concentration in the bottoms, which are protected against any oxidizing influences by the supernatant sulphides. This condition, very likely, does not exist to any great extent in the converter, where the strong blast is more apt to keep the copper and the sulphide in such whirl and ebullition as to make both of them offer large surfaces for the attack of oxygen.

Lead is about equally eliminated by both methods. This is readily understood when we recall the facts that it is both slaggable and volatile, and exercises no choice between copper and sulphide, thus offering itself constantly to the oxidizing action.

Arsenic shows a much wider difference of behavior in the two processes than the foregoing elements. The slight degree of elimination in the reverberatory-process has been accounted for as due to the high degree of concentration in the copper-bottoms, exceeding considerably that of any other element. On the other hand, it has been shown that in the calcining of matte the amount of arsenic eliminated is many times greater than that of any other elements. Matte-calcining, however, is a very mild operation compared with the most severe oxida-

tion, which takes place at a very high temperature in the converter. It is more than probable that by far the largest part of the arsenic is volatilized in the converter during the period previous to the formation of any metallic copper. When once combined with this metal, arsenic proves to be far more refractory than when in the sulphides.

Antimony somewhat evades elimination by concentration in the copper-bottoms of the reverberatory-process. In both processes, however, it is a more slaggable element than any of the others under discussion.

Selenium and tellurium are volatile elements, and, since they concentrate in the sulphides, are exposed to oxidation. Apparently they are carried into the slags mainly by metallic copper. No plausible reason can be offered for the great differences in the degree of elimination of these elements by the several converters.

It would extend the volume of this paper much too far to describe in detail the methods by which the elements were determined in these experiments. Suffice it to say, therefore, that the same method for each element was consistently and carefully carried through with every analysis by the writer himself.

It may be added that for the purpose of determining those elements in the slag which are considered as impurities in the metallic copper, the slags (in quantities from 20 to 300 grammes) were fused with an ordinary fusing-mixture, a reducer, and, for those of small copper-contents, with an addition of copper oxides (obtained by calcining filings of the purest electrolytic copper). The resulting copper-bead was analyzed in the same manner as other metallic copper, and very concordant results were obtained.

The Ultimate and the Rational Analysis of Clays and Their Relative Advantages.

BY HEINRICH RIES, PH.D., NEW YORK CITY.

(Atlantic City Meeting, February, 1898.)

In another place,* the writer has called attention to the modern methods of the laboratory investigation of clay, and it is desired here simply to discuss one branch of the subject, which, though of considerable importance, has heretofore received little attention in this country, viz., the rational analysis of clay. In order, however, the better to point out the bearings and advantages of this procedure, it is necessary to refer briefly to the ultimate analysis as well. The remarks in this paper refer only to the use of clay for the manufacture of clay-products.

Pure clay or kaolin† is extremely refractory, possesses little plasticity, and usually shrinks and warps considerably in burning. Even the smallest proportion of impurities changes these properties, so that the kaolin exhibits variations in shrinkage, plasticity, tensile strength, fusibility, and often color when burned. The majority of nearly pure kaolins consist of kaolinite (Al_2O_3, $2SiO_2$, $2H_2O$), undecomposed feldspar, and quartz. The kaolinite generally forms the finest particles of the clay and is known as the clay-substance, while the rest of the clay-particles are collectively spoken of as sand.

In the ordinary or ultimate analysis, the substances generally determined are silica, alumina, ferric oxide, lime, magnesia, alkalies and water. An analysis of this type simply regards the clay as a mixture of elements or oxides, but gives no clue as to the actual condition in which these substances exist, *i.e.*, whether as silicates, oxides or carbonates, etc., a point which it is of the highest importance to know; for the individual components of the clay influence its properties not only according

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* 18th *Ann. Rep't. U. S. Geol. Surv.*, Part V. (continued), p. 1122.

† The term kaolinite refers to the mineral resulting from decomposition of feldspar, while kaolin is used to designate a mass of more or less pure kaolinite. The two names are sometimes confused.

to their quantity, but also according to their condition. Thus, lime, if present as a carbonate, may be extremely injurious, causing the clay to swell or burst during or after burning, while if present as a silicate (feldspar) it may form a desirable flux, serving to bind the burned clay-particles more firmly together.

Moreover, the ultimate analysis gives no clue to the condition in which silica is present in the clay. In the form of quartz it would serve to diminish the shrinkage, and (except at high temperatures) increase the refractoriness of the clay, but if combined with alumina and potash in feldspar it would act as a fluxing agent.

A high percentage of total silica in an ultimate analysis does not necessarily indicate the presence of much quartz, but may be due, on the contrary, to an excess of feldspar.

Regarding the physical properties of the kaolin or clay, the ultimate analysis gives us few clues. Certain inferences, however, may be drawn from it. Pure or nearly pure clay burns white, but iron tends to develop a red color, the intensity of which increases with the temperature to which the clay is burned, and the amount of iron present. Small percentages of iron-oxide will impart only a yellow or yellowish-white tint, while large quantities may give deep red. (This statement assumes that in each case the clay is burned in an oxidizing atmosphere. A reducing one would give other colors.)

If both iron and lime exist in the clay, we can see from the ultimate analysis whether the proportion of lime to iron is such as to color the burned clay yellow or yellowish-green.

Still another inference may be drawn from the ultimate analysis, namely, as to the degree of refractoriness possessed by the clay. It may be said, in general, that, other things being equal, the greater the percentage of fluxing impurities, such as iron, lime, magnesia and alkalies, the greater will be the fusibility of the clay.

The following analyses indicate this fact:

	1. Per cent.	2. Per cent.	3. Per cent.
SiO_2,	47.20	69.50	54.90
Al_2O_3,	36.50	13.00	18.03
Fe_2O_3,	2.56	6.40	6.03
CaO,	tr.	.25	2.88
MgO,	tr.	tr.	1.10

	1. Per cent.	2. Per cent.	3. Per cent.
Alkalies, ...		tr.	3.40
H_2O, ...	13.35	6.70	6.90
Moisture, ...	.50	3.40	3.17
Total fluxes, ...	2.56	6.65	13.41
	Deg. F.	Deg. F.	Deg. F.
Viscosity or fusion point, . above	2700	2300	1900

1. Chalk Bluff, Marion Co., Ala., *U. S. Geol. Surv.*, 18th *Ann. Rep.*, Part V. (continued), p. 1128.
2. Fernbank, Lamar Co., Ala. *Ibid.*
3. Norborne, Mo. *Mo. Geol. Surv.*, XI. *Ann. Rep.*

This is practically the full extent to which the ultimate analysis can be used; and there still remain to be explained a number of physical facts concerning any clay which happens to be under consideration.

It frequently happens that two clays approach each other quite closely in their ultimate composition, and still exhibit an entirely different behavior when burned. The explanation which most quickly suggests itself is, that the elements present in the two clays are differently combined. Some method of resolving the clay into its mineral components, so as to indicate the condition in which the elements are present is therefore practically needed.

As kaolinite results from the decomposition of feldspar, the kaolin is quite sure to contain some undecomposed feldspar, and also some quartz, and (in smaller amounts) mica, since the two latter minerals are common associates of the feldspar.

If, now, we know the amount of feldspar, quartz and kaolinite or clay-substance in the kaolin, and the effect of these individual minerals, we can form a far better opinion of the probable behavior of the clay in burning.

The rational analysis, or method of separating kaolin into its mineralogical components, depends on the decomposition and solution of kaolinite by sulphuric acid and sodic hydrate, leaving the feldspar and quartz behind.* In this residue the alumina is determined; and from this the amount of feldspar in the residue can be calculated. The balance is quartz. When mica is present, it is dissolved out with the kaolinite and reckoned in as clay-substance, but it is rarely present in large amounts,

* The rational analysis is in reality a simple operation, but still requires some little practice to ensure accurate results.

and may perhaps alter the character of the clay-substance but little, for finely ground white mica possesses plasticity, and can be formed and dried without cracking. It is more refractory than feldspar, and holds its form up to 1400° C.*

In the table on the following page are given the ultimate and rational analyses of a number of kaolins, which show how a constancy of ultimate composition may be accompanied by variations in the rational analysis.

From this table a number of interesting conclusions may be drawn. Columns 1 and 2 represent two clays which agree very closely in their ultimate composition; but in the rational analysis there is a difference of 6 per cent. in the clay-substance, 12 per cent. in the quartz, and nearly 19 per cent. in the feldspar. Nos. 3 and 5 and 10 and 12 also illustrate this point.

In Nos. 6 and 7, one a German, and the other a North Carolina kaolin, the ultimate analyses are very closely alike, and the rational analyses also agree very well. This is frequently the case when the clay-substance is very high, between 96 and 100 per cent., as in Nos. 9 and 11.

A third case would be presented if the rational analyses agreed, but the ultimates did not. Such instances, however, seem to be much less common.

The practical value of the rational analysis bears chiefly upon those branches of the clay-working industry, such as the manufacture of porcelain, white earthenware, fire-bricks and glass-pots, which use materials with comparatively few fusible impurities (iron, lime, magnesia).

There is much concerning clays which still remains unexplained, but it seems probable that, other things being equal, two clays having the same *rational* composition will behave alike.

We can illustrate this point by the following tests made on washed kaolins from the vicinity of Sennewitz, near Halle, Germany. From the figures given below, it will be noticed that in the case of Nos. 1 and 2 there is a close agreement in the shrinkage, which amounted to about 10 per cent. when the clay was heated up to the temperature of a hard-porcelain

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* G. Vogt, *Chem. News*, Dec. 26, 1890, p. 315.

TABLE I.—*Ultimate and Rational Analyses of Clays.*

	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.	12.
						ULTIMATE ANALYSIS.						
SiO_2	62.40	62.52	63.17	64.87	63.07	51.51	53.10	47.60	46.61	58.39	46.82	57.08
Al_2O_3	26.51	25.57	25.09	23.83	24.67	31.41	33.06	34.00	36.47	27.52	38.49	29.94
Fe_2O_3	1.14	.92	.64	.83	.59	.68	1.18	1.30	2.81	.36	1.09	.65
CaO	.57	.65	.35	....	....	.04	.38	Tr.	.14	1.52	....	....
MgO	.01	.10	.26	.50	.40	.43	.08	.50	....	.41	Tr.	.49
Alkalies	.98	1.04	.80	1.39	4.25	.55	.83	3.00	1.44	4.29	1.40	2.26
Loss by ignition	8.80	9.27	9.70	8.36	7.00	12.37	11.32	13.60	12.80	7.19	12.86	9.87
	100.41	100.07	100.01	99.78	99.98	99.99	99.95	100.00	100.27	99.68	100.66	100.29
						RATIONAL ANALYSIS.						
Clay-substance	66.33	72.05	67.82	63.77	54.92	83.04	83.39	88.34	96.08	55.88	96.55	74.09
Quartz	15.61	27.78	30.93	35.50	23.52	16.28	14.99	8.95	1.93	5.95	2.30	17.21
Feldspar	18.91	.10	1.25	.73	21.56	.68	1.57	2.73	1.99	38.17	1.15	8.70
	100.85	99.93	100.00	100.00	100.00	100.00	99.95	100.02	100.00	100.00	100.00	100.00

1. Crude kaolin, Springer mine, Webster, N. C. *Bull. N. C. Geol. Surv.*, on "Clays of North Carolina."
2. Slip clay from Rühle's mine, Löthain, Saxony. *Thon-Ind.-Zeit.*, 1892, p. 1031.
3. Slip-clay from Kaschkau, Germany. *Ibid.*
4. Kaolin from Sennewitz, Saxony. *Notizblatt*, 1876.
5. Porcelain-clay mixture. *Ibid.*
6. White earthenware-clay, Löthain, Saxony. *Seger's Ges. Schr.*, p. 887.
7. Kaolin (unwashed), West Mills, N. C. *Bull. N. C. Geol. Surv.*, on "Clays of North Carolina."
8. White earthenware clay, Wiesau, Germany. *Thon-Ind.-Zeit.*, 1894, p. 358.
9. Fire-clay, Bautzen Germany. *Ibid.*, 1894, p. 842.
10. Kaolin, Limoges, France. *Seger's Ges. Schr.*, p. 552.
11. Kaolin. Zettlitz, Bohemia. *Ibid.*, p. 50.
12. Kaolin, Lettin, Saxony. *Ibid.*, p. 50.

kiln. In Nos. 3 and 4 the shrinkage is very nearly the same, but greater than in Nos. 1 and 2, because the rational composition has changed, there being a marked increase in the amount of feldspar.

If there had been much difference in the size of the clay-particles of Nos. 3 and 4 or Nos. 1 and 2, the shrinkage in each case would probably have been different.

TABLE II.—*Rational Analysis and Shrinkage of Clays.*

Feldspar. Per cent.	Quartz. Per cent.	Clay-Substance. Per cent.	Fe_2O_3. Per cent.	Shrinkage in Hard Porcelain Fire. Per cent.
1.59	33.86	64.55	0.75	10.20
1.21	33.39	65.40	0.73	10.10
8.64	31.69	59.68	0.30	12.90
8.25	35.15	56.60	0.30	12.00

The degree of fineness of the clay-particles, and perhaps their shape also, probably exert more influence on the shrinkage than has been imagined, but just how far this makes itself felt is still undetermined.

As an illustration of the practical use of the rational analysis we may take the following:

Suppose that we are using for the manufacture of porcelain or fire-brick a kaolin which has 67.82 per cent. of clay-substance, 30.93 of quartz, and 1.25 of feldspar, and that to 100 parts of this is added 50 parts of feldspar. This would give us a mixture of 45.21 per cent. of clay-substance, 20.62 of quartz, and 34.17 of feldspar.

If now for the clay we had been using, we substituted one with 66.33 per cent. of clay-substance, 15.61 of quartz, and 18.91 of feldspar, and made no other changes, the mixture would then contain 44.22 per cent. of clay-substance, 10.41 of quartz and 45.98 of feldspar.

This last mixture shows such an increase in feldspar that it must give much greater shrinkage and fusibility; but knowing the rational analysis of the new clay, it would be easy to add quartz or feldspar so as to bring the mixture back to its normal composition.

The application of the method of rational analysis to impure clays is not quite as satisfactory, but at the same time not as necessary. In the treatment, the iron, if present as oxide, and

lime or magnesia, if carbonates, are dissolved out with the clay-substance. The silicate minerals are grouped with the feldspar, and the clay thus becomes divided into clay-substance (kaolinite, ferric oxide, lime and magnesia carbonates); feldspar or feldspathic detritus; and quartz. If the percentage of ferric oxide and carbonates is high, it is necessary to determine them separately in the ultimate analysis.

The importance of the rational method of analysis has been widely recognized in Europe, especially in Germany, and has been applied with great success; and it is hoped that its value will soon become more thoroughly appreciated in this country. The complete investigation of a clay in the laboratory should include not only an ultimate and a rational analysis, but also detailed physical investigation.

An Automatic Feed-Device for Gas-Producers.

BY C. W. BILDT, WORCESTER, MASS.

(Atlantic City Meeting, February, 1898.)

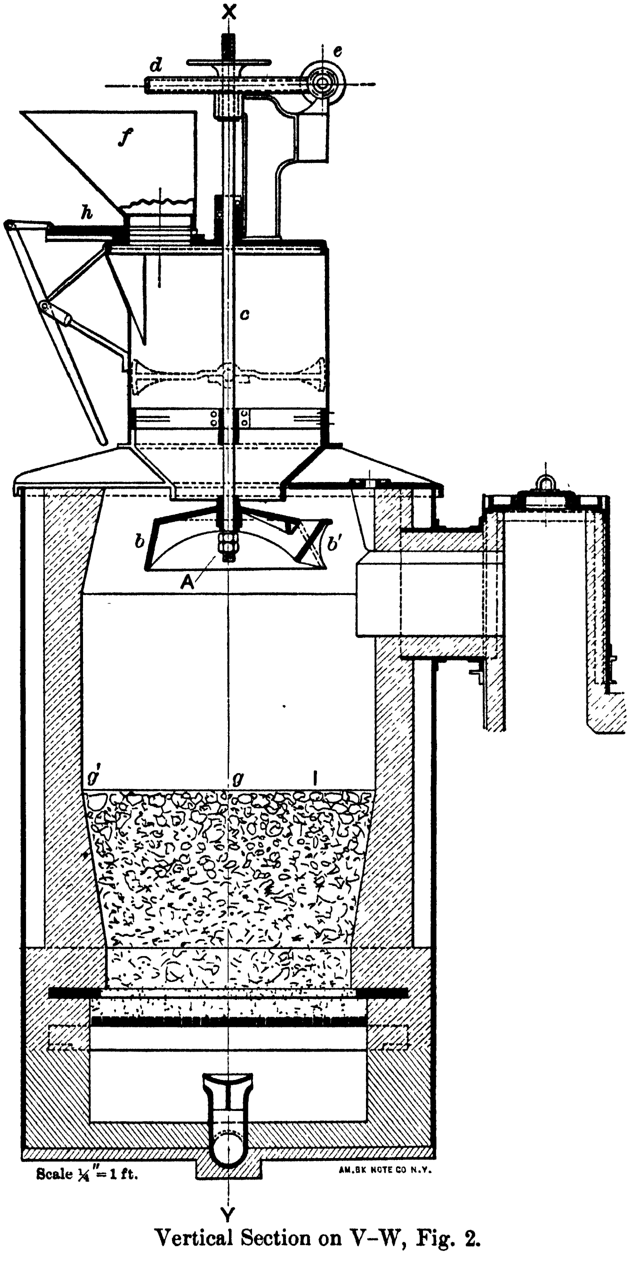
During many years of service in the iron and steel industry I have frequently found, as have also many other engineers, that the common devices used for feeding coal into gas-producers are not what they ought to be in order to secure the best economy.

The hopper-and-cone so generally used is familiar to all, and a brief study of its action will surely satisfy everybody that this device has several serious faults.

In the first place, the charging is intermittent, and depends entirely upon an attendant. When the hopper is frequently emptied, it may work fairly well, but even then it suddenly discharges into the hot producers a large body of coal, which is often violently converted into gas that rushes into the furnace, and, as the damper for the combustion-air cannot be regulated every time the producer is charged, there will be an excess of gas, which goes up the chimney unburned.

Between the chargings the quantity of gas generated decreases, and there will often be an excess of air in the furnace,

FIG. 1.

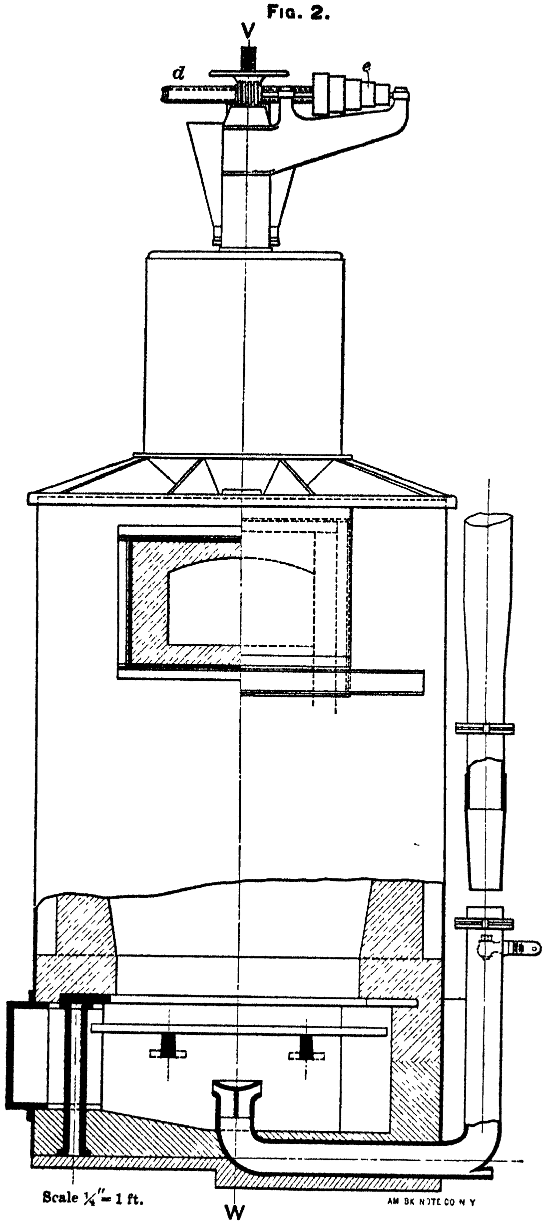


Vertical Section on V–W, Fig. 2.

which carries off the heat and causes oxidation of the metal. In either case there is a loss of fuel.

The cone will never distribute the coal evenly over the

FIG. 2.

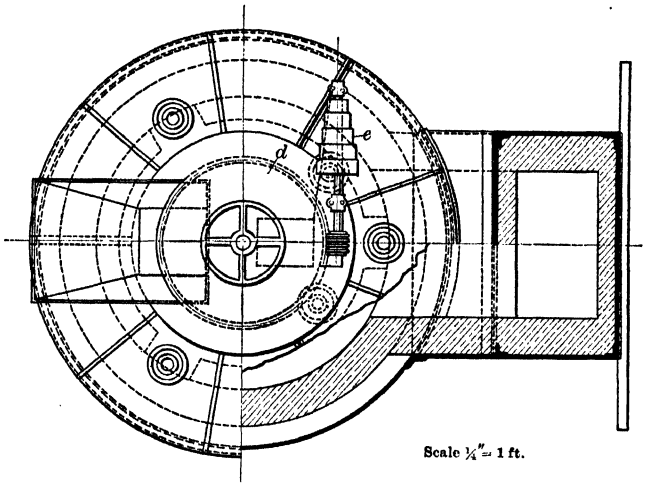


Vertical Section on X–Y, Fig. 1.

charging-surface in the producer. It will sometimes deposit more on one side than on the other, or more on the sides than in the middle, resulting in a body varying in thickness, through

which the air flows where it meets the least resistance. The result of this is a poorer gas, often carrying an excess of car-

FIG. 3.



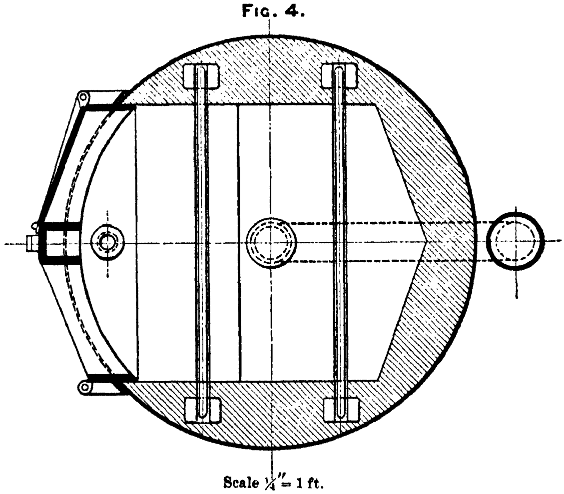
Scale ¼″ = 1 ft.

Top View (Fig. 1).

bonic acid, and continually varying in its chemical composition.

This defect of uneven distribution can, of course, be remedied

FIG. 4.



Scale ¼″ = 1 ft.

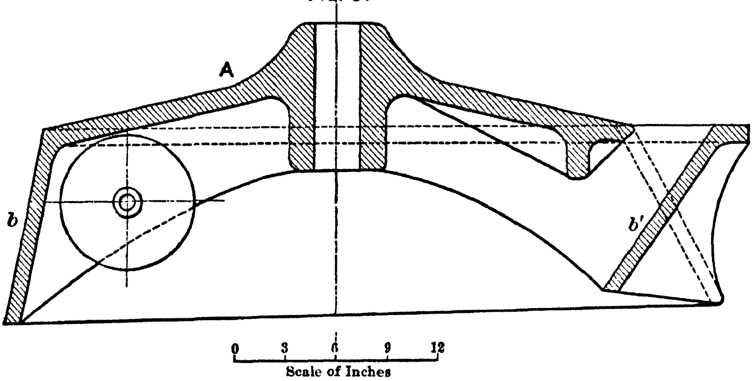
Ground Plan (Fig. 2).

to a certain extent by poking, but that is not a satisfactory remedy, and involves hard labor, which the attendant often shirks as much as possible.

Remembering these defects, we easily find that an ideal feed-device should possess the following features:

1. It should be continuous, steadily supplying during a certain period of time an amount of coal that will produce the amount of gas required during the same length of time.

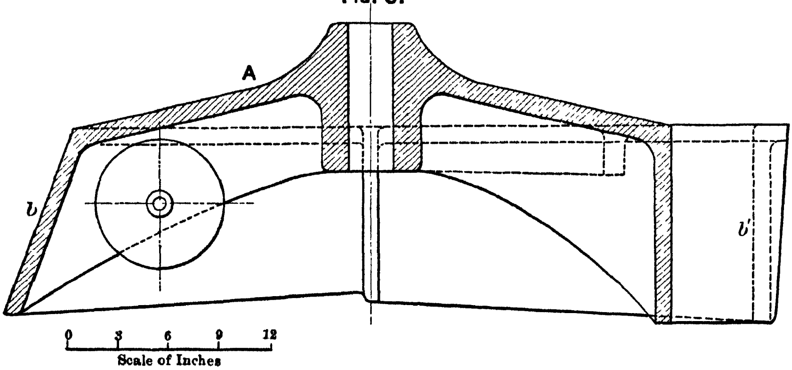
FIG. 5.



Enlarged Section of A, *b*, *b′*, Fig. 1, on line 1—7, Fig. 7.

2. It should distribute the coal uniformly over the surface in the producer, making a body of equal thickness, throughout which the air meets with an equal resistance. It should also

FIG. 6.



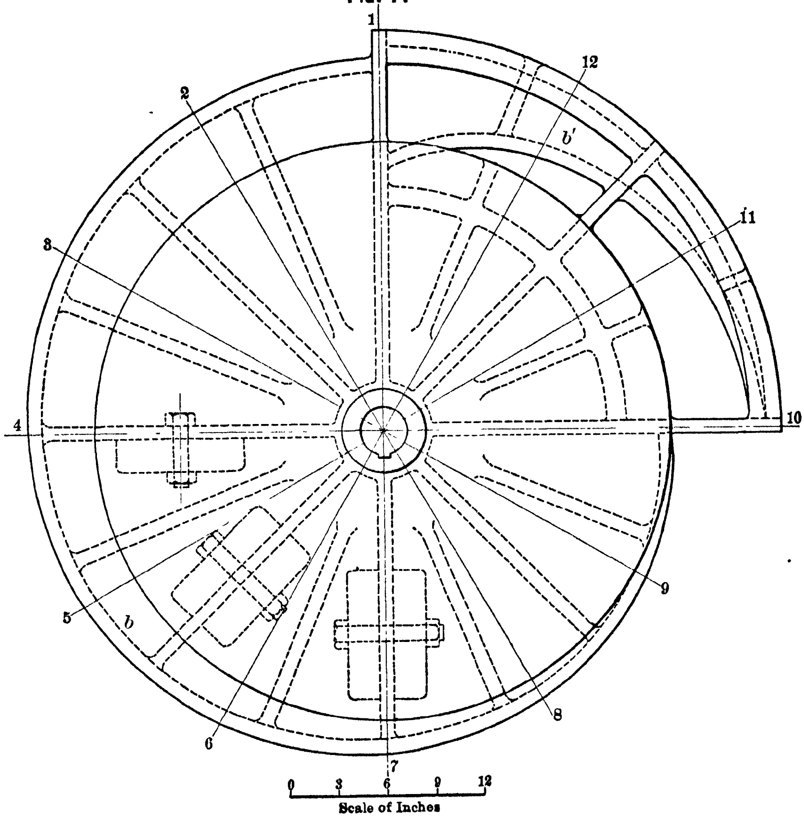
Enlarged Section of A, *b*, *b′*, Fig. 1, on line 4—10, Fig. 7.

be simple in its construction, needing very little repairs, so that it may be run by unskilled labor.

The author believes that the feed-device constructed and patented by him, and already put to long and severe tests in several places, will meet the above requirements.

It consists, as will be seen from Figs. 1 to 7, of a coal-holder resting upon and attached to the top of the producer, below which is placed a rotating distributing-disk A, provided with specially-constructed blades *b* and *b'*, serving to distribute the coal evenly over the charging-area I. The disk A is rotated by the shaft *c* resting upon the top of the coal-holder,

FIG. 7.



Enlarged Horizontal Projection of A, *b*, *b'*, Fig. 1.

and the worm-gearing *d*, which, in its turn, is driven by cone-pulleys *e*, by which the speed can easily and quickly be changed to feed more or less coal, as may be required. To suit different sizes of coal, the opening between the discharging-mouth of the holder and the disk is adjusted by lifting the vertical shaft, which slides through the worm-gear. This may be done by a screw-wheel, as indicated in the drawing, or by any other suitable arrangement.

The size of the coal is immaterial so far as the charging is concerned; lumps, dust, or both mixed, are equally well distributed.

The top of the rotating-disk can be made level or sloping (not exceeding the angle of friction between the disk and the

FIG. 8.

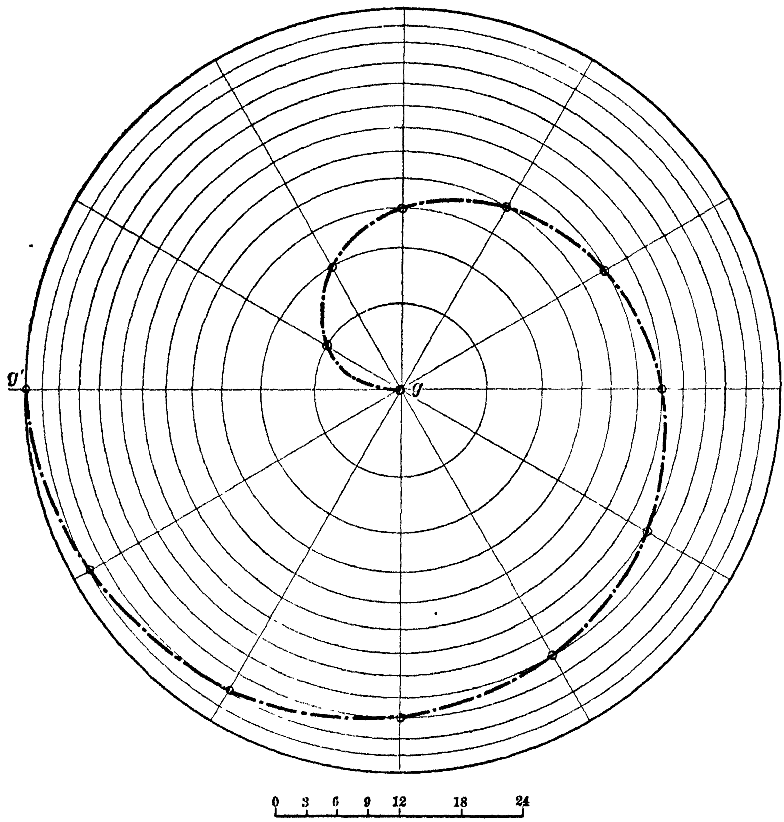


Diagram Showing Construction of Spiral.

coal). The speed of the disk is about one revolution in four minutes.

The holder, which can be made large enough to receive any desired quantity of coal, is filled through hopper *f* and sliding-damper *h* from any larger deposit supplied by a coal-conveyor, or by any other convenient method.

As the distributing-blades *b b'* are the most important part of the apparatus, I wish to explain how they are constructed.

The action of the apparatus is based upon a spiral, according to which the coal sliding down the blades is distributed upon the surface below.

The spiral goes from the periphery to the center of the charging-area, and is constructed as follows: Divide the surface of the producer into any number of annular rings of equal area by drawing concentric circles. Then divide the same surface by radii forming equal angles into as many parts as there are rings, and by a line connect the points of intersection between the circles and the radii, as shown in Fig. 8.

When this spiral has been constructed of such dimensions that the distance from the center, g, of the circles to the extreme point, g', on the curve is equal to the radius $g\ g'$ of the circular charging-surface of the producer, the distributing-blades or flanges can be adjusted to the charging-disk A. As, at the same time, equally large quantities of coal pass equally large divisions on the surface of the disk, it follows that those parts of the spiral intersected by equally large angles, with their vertices directed toward the rotary shaft of the apparatus, receive equally large quantities of coal during the same period. If we now construct the distributing-curve in such way that these equal parts shall, in their motion around the axis, at any moment, cover equal surfaces of producer, the coal will naturally be evenly distributed. The parabolas which direct the coal in its discharge from the lower edge of the blade to the surface must be taken into consideration in the construction of the distributing-blades. As the blades are arranged on the drawing, the blade b distributes the coal over the surface outside of the distributing-disk A, and the blade b' discharges the coal directly under the disk A.

The apparatus can be applied to producers of any size, as it distributes the coal equally well over a large or small area.

A large producer should, therefore, be used. It is cheaper to build, takes less room, requires less labor in tending, and the loss of heat by radiation is smaller than in several producers with corresponding aggregate area.

In order to obtain the best results it is important that the producer be proportioned to the furnace which it shall supply. In practice I have found that an open-hearth furnace requires 3½ square feet of grate-area per ton of capacity.

A 10-ton open-hearth furnace, for instance, would need a producer with 35 square feet of grate-area.

For heating-furnaces 5 square feet of grate-surface is allowed per ton of steel heated per hour. A furnace heating 40 tons per 10 hours would require a grate-area of 20 square feet.

These dimensions depend, of course, somewhat on the material to be heated and the quality of the coal. When using the producer for a heating-furnace it is important that the connection or gas-flue be as short as possible, in order to save heat, as the gas is not regenerated, and that the gas be taken out as near the top of the producer as possible, to prevent any coal-dust going over into the furnace.

The top-blast in the heating-furnace should be heated, which can be done cheaply by forcing the air through brick flues under the furnace-bottom.

This apparatus can, of course, be applied to any style of producer, although I have shown in the drawings a well-known type, with plane grate and steam-blast, to which I give preference.

This feed-device is now used, and has been used for some time, at the following places, where it has given excellent results:

At the Stridsberg & Bjorck works, Trollhattan, Sweden, one producer has been operated in connection with an open-hearth furnace during three years. The saving of coal is 15 per cent. over the old type.

Stora Kopparbergs Bergslag's work, Domnarfvet, Sweden, uses three producers combined with heating-furnaces: One with 12½ square feet grate-area, for heating 4-inch billets, in use 17 months; one with 35 square feet grate-area, for heating ingots, in use 11 months; one with 20 square feet grate-area, for heating plate-ingots, in use 4 months.

At Boxholm Iron-Works, Sweden, one producer of 12½ square feet grate-area has been operated 16 months for heating ingots and charcoal-blooms.

At the Avesta Iron-Works, Sweden, one producer, used for heating ingots, has been used 11 months.

At Carlsvik, Stockholm, Sweden, one producer is running in connection with a malleable-iron furnace.

At the E. Bocking Walswerken, Mulheim am Rhein, Germany, one producer runs a heating-furnace.

At the Washburn & Moen Mfg. Co.'s works, Worcester, Mass., one producer has been run for the last 6 months in connection with a heating-furnace. The grate-area is 20 square feet, and the capacity of the furnace is 40 to 50 tons of 4-inch billets per 10 hours, charged cold.

From these works I have received the following testimony in regard to the working of the producers:

1. The gas is of an excellent quality, steady in its flow, making a long flame which preheats the material thoroughly. The analysis is as follows:

	From Domnarfvet. Per cent.	Washburn & Moen Mfg. Co. Per cent.
CO_2,	2.00	4.90
O,	0.10	None
CO,	27.90	26.80
C_2H_4,	0.10	0.40
H,	2.60	18.10
CH_4,		3.50
N,	67.60	46.30
	100.30	100.00

The value of steam-blast under the producer-grate is here strikingly shown. The first sample is taken from a producer which is run principally by air-blast, with a little steam mixed into the air. The second sample is taken from a producer where the air is forced in by the steam on the injector principle.

2. The consumption of coal per ton of finished material has decreased 15 to 30 per cent., and at one of the works, which run with the comparatively low coal-record of 180 to 200 pounds per gross ton of blooms heated (charged cold), my device lowered the consumption to 142 pounds per gross ton.

3. As a result of the steady and uniform flow of the gas, the furnace-waste has been considerably lowered—a factor more important than the saving of coal. At one of the works mentioned the waste from 4-inch blooms to rods is reduced to 1.65 per cent. The blooms, or ingots, are well-heated, with smooth corners, show none of the coal- or sulphur-spots, so often produced by old furnaces with fire-boxes.

4. Labor has been reduced to a minimum at the producer. No poking is required, and no clinkers are formed, because of the uniform heat in the body of coal. The residue is only

clear ashes, which are easily cleaned out by shaking the grates or running a bar just above them.

5. The producer can, therefore, be quickly started on Monday morning, and in about 2½ hours the furnace is at welding-heat.

6. As no cooling-down is required, the amount of repair is exceedingly slight, both in the producer and in the furnace, and furnaces and producers have been run for several months without the slightest repairs.

7. The distributing-disk shows no wear, and, on account of the low temperature in the producer, is not burnt out. At Trollhattan the same disk which was put in three years ago is still running, and the same is the case at the other works using my device. This will show that water-cooling of the feed-device is not necessary.

All the testimonials received may be summed up in a few words: Gas of uniform and excellent quality, steady supply, increased furnace-capacity, lower coal-consumption per ton of material heated, and reduced furnace-waste, labor and repairs.

This feed-device is covered by U. S. Letters Patent No. 442,676, issued to me, December 16, 1890, and No. 498,229, issued to me May 30, 1893.

The Influence of Antimony on the Cold-Shortness of Brass.

BY ERWIN S. SPERRY, BRIDGEPORT, CONN.

(Atlantic City Meeting, February, 1898.)

THE formation of cracks in metals is one of the most perplexing obstacles encountered during the process of rolling. When occurring in brass they may be due to several causes:

1. Shrinkage-cracks. These are caused by a rough or cracked mould which prevents the metal from shrinking uniformly and without rupture. It is consequently torn apart. In the majority of instances such cracks are visible to the naked eye, and often penetrate the metal to a considerable depth; in others they are nearly if not quite microscopic, but

become apparent when an attempt is made to roll the alloy. The preventive of shrinkage-cracks is a smooth mould.

2. Fire-cracks formed during the annealing. The cause of this difficulty is either insufficient "breaking-down" to equalize the compression or the absence of "whipping" before annealing.

3. Rolling-cracks. The cause of such cracks is difficult to determine, as it is inherent in the metal. They occur at two different stages of the process, before and after the first annealing. When they occur after the first annealing and have not appeared during the "breaking-down," they are usually due to imperfect annealing, provided a normal reduction has been given. When they occur while the material is being "broken-down," they may be due to one or more of the following causes:

a. Too great a reduction.

b. Induced crystallization on account of the alloy being excessively hot when poured.

c. Imperfect mixing of the ingredients.

d. Alloy too "high" to roll with the standard reduction.

e. Foreign elements in the alloy.

The first four causes are well known, and the difficulty they entail is easily remedied; but the influence of foreign elements on the rolling has been imperfectly understood. Nearly all the elements have been charged with injuring brass, but no experiments seem to have been made to bear out these assertions.

The frequent cracking of brass in "breaking-down" when both the mixture and the amount of reduction are normal indicates that the impurities in the copper are the cause of the trouble; and since it is experienced chiefly when electrolytic copper is employed, a synthetic investigation of the effect produced upon brass by the principal impurities occurring in electrolytic copper seems to be fundamental to the inquiry into this cause of cracking. Antimony being, probably, the principal impurity in such copper, this element was selected as the first to be thus studied.

Varying amounts of antimony were added to brass of a constant composition, and the products were tested by rolling and by examination of the fracture of the cast material. For the base an alloy containing 60 per cent. of copper and 40 per

cent. of zinc was used. This mixture was selected for the reason that it is practically the hardest copper-and-zinc alloy that will roll cold, and, therefore, the influence of any impurity is more apparent than it would be if a softer mixture were used. Again, this alloy is a "high" brass composition which will roll both cold and hot.

The purest Lake copper and refined zinc were used, and the antimony was introduced as an alloy of copper and antimony. The copper was melted in a plumbago crucible under a sufficient layer of charcoal, and the alloy of copper and antimony was then introduced; the zinc was next added, and the mass was stirred with a plumbago stirrer and allowed to attain a temperature suitable for pouring. Great care was taken to "spelter" at the lowest possible temperature, so as to avoid overheating the copper. The molten material was next poured into a mould $\frac{5}{8}$ by $2\frac{5}{8}$ by 24 inches, which had been heated and coated with sperm oil. The usual precautions were taken to surround the stream of molten metal with gaseous hydrocarbons to prevent the formation of oxide. The plate was scraped thoroughly before and after "breaking-down." All rolling was performed cold.

Experiment No. 1.

Copper, 59.46; zinc, 39.87; antimony, 0.67 per cent. Melted 5.5 pounds of copper, added 3 pounds 11 ounces of zinc, and then 1 ounce of antimony. Rolled from 0.468 to 0.431 inch, a reduction of 8 per cent. Many cracks started transversely on both sides of the plate. Several of them were quite deep, and could not be removed by scraping. Upon further rolling (without annealing) to 0.410 inch, a total reduction of 12 per cent., the plate cracked to pieces.

It is evident from this experiment that the alloy employed will not roll cold; and the fact is emphasized by the appearance of the fracture, which is exceedingly short and crystalline. The alloy is considerably harder than the same composition without antimony, and the color is not as yellow, but of a brownish tint. It could not be forged at any temperature, but cracked to pieces. The appearance of the fracture is shown in Fig. 1.

Experiment No. 2.

Copper, 60; zinc, 39.90; antimony, 0.10 per cent. Melted 5 pounds of copper, added 1.33 ounces of an alloy of 90 copper to 10 antimony, and then 3 pounds 6 ounces of zinc. Rolled from 0.655 to 0.560 inch, a reduction of 14 per cent. Checked badly on both sides of the plate, but no edge-cracks appeared. Annealed and rolled to 0.476 inch, a reduction of 15 per cent. Although, after the previous reduction, the checks were apparently removed by annealing, they reappeared again, after this reduction, to an enormous extent; so much so, in fact, that it was almost impossible to remove them. Annealed and removed as many of the checks as possible by scraping, and then rolled to 0.385 inch, a reduction of 17 per cent. No edge-cracks had yet appeared; but the checks had increased in number and depth, becoming veritable cracks. Annealed and rolled to 0.288 inch, a reduction of 25 per cent. and so many cracks formed, and the old ones became so extended, that it was impossible to roll any further.

The fracture of this alloy, while tougher than that of the preceding one, is distinctly crystalline. It is shown in Fig. 2. The alloy could be forged hot to a thin edge, and bent over on itself and flattened, without showing any cracks.

Experiment No. 3.

Copper, 59.97; zinc, 39.98; antimony, 0.05 per cent. Melted 5 pounds 15.25 ounces of copper; added 0.8 ounce of an alloy of copper 90 per cent. and antimony 10 per cent., and then 4 pounds of zinc. Rolled from 0.625 to 0.540 inch, a reduction of 13 per cent. Checked on both sides of the plate, but did not crack on the edges. Annealed, scraped out the checks and rolled to 0.459 inch, a reduction of 15 per cent. Checked badly again, but no cracks appeared on the edges. Annealed and rolled to 0.334 inch, a reduction of 27 per cent. Checked so badly during this reduction that it became nearly impossible to roll the sheet any further. No cracks, however, had yet appeared on the edges. Annealed and rolled to 0.256 inch, a reduction of 23 per cent. Checks appeared again in large numbers, although they had been partially removed before this reduction. No edge-cracks formed. Annealed and rolled to 0.168 inch, a reduction of 33 per cent. Checks still increased in size

and number, but the sheet did not crack on the edges. The appearance of the surface was quite rough, due to the checks being rolled down. Annealed, scraped and rolled to 0.080 inch, a reduction of 52 per cent. A few cracks appeared on the edges during this reduction, but the checks had ceased to form, those which previously existed having been rolled into the metal. Annealed and rolled to 0.048 inch, a reduction of 40 per cent. Did not crack on the edges during this reduction. The appearance of the sheet at this stage was "spilly," but the following test was made on as smooth a section as it was possible to obtain:

Annealed Sheet.—Length, 12 inches; dimensions, 0.048 by 0.498 inch; sectional area, 0.0239 square inch; tensile strength, 1350 pounds, or 56,600 pounds per square inch; elongation: in 1 inch, 26 per cent.; in 8 inches, 21 per cent.; reduction of area, 27 per cent.

The fracture of this alloy, as shown in Fig. 3, is distinctly crystalline, and the crystals are quite coarse.

Experiment No. 4.

Copper, 60.06; zinc, 39.92; antimony, 0.02 per cent. Melted 4.5 pounds of copper, added 0.25 ounce of an alloy of 90 copper and 10 antimony, and then 3 pounds of zinc. Rolled from 0.605 inch to 0.528 inch, a reduction of 12 per cent. No cracks or checks appeared. Annealed and rolled to 0.332 inch, a reduction of 37 per cent. No cracks appeared on the edges, but a large number of checks formed on the surface. Annealed, removed the checks by scraping and rolled to 0.130 inch, a reduction of 60 per cent. No more checks appeared, but a few cracks formed on the edges. Annealed and rolled to 0.046 inch, a reduction of 64 per cent. No checks appeared during this reduction, but the sheet cracked considerably on the edges.

The amount of antimony in this alloy seems to represent the dividing line between good and bad material. Although the alloy rolled without much difficulty, the tendency which it manifested to check would undoubtedly cause more or less trouble in practice. The fracture, shown in Fig. 4, betrays the cause of the trouble. The difference in appearance between the fractures shown in Figs. 4 and 5 is worthy of notice.

The following test was made on the sheet rolled from this alloy:

Annealed Sheet.—Length, 12 inches; dimensions, 0.046 by 0.497 inch; sectional area, 0.0228 square inch; tensile strength, 1348 pounds, or 58,800 pounds per square inch; elongation. in 1 inch, 52 per cent.; in 8 inches, 34.5 per cent.; reduction of area, 45 per cent.

This alloy forged hot as well as the previous alloy, or even as the same composition without antimony.

Experiment No. 5.

Copper, 62.02; zinc, 37.96; antimony, 0.02 per cent. Melted 6 pounds of copper, added 3.25 ounces of an alloy of 99 copper and 1 antimony, and then 3 pounds, 12.75 ounces of zinc. Rolled from 0.593 inch to 0.492 inch, a reduction of 17 per cent. Cracked badly on the edges. Annealed and rolled to 0.340 inch, a reduction of 30 per cent. Checked all over the surface on both sides, and cracked on the edges considerably. Annealed and removed the checks as far as practicable by scraping, and rolled to 0.225 inch, a reduction of 33 per cent. Checked badly again, but the cracks on the edges did not increase in size or number. Annealed, scraped and rolled to 0.054 inch, a reduction of 76 per cent. Cracked badly on the edges, but most of the checks had disappeared, having been rolled in.

The fracture of this alloy is crystalline, and cannot be distinguished from that of Experiment No. 4. (See Fig. 4.)

The alloy would forge hot to a thin edge without cracking, but could not be bent even 45 degrees without a fracture.

This experiment was made in order to find the effect of antimony on a lower composition than that of Experiment No. 4. The effect on these two alloys is identical.

The following tests were made on a sheet rolled from the above alloy:

Annealed Sheet.—Length, 12 inches; dimensions, 0.054 by 0.485 inch; sectional area, 0.02619 square inch; tensile strength, 1410 pounds, or 53,800 pounds per square inch; elongation: in 1 inch, 53 per cent.; in 8 inches, 43 per cent.; reduction of area, 58 per cent.

Hard-Rolled Sheet.—Length, 12 inches; dimensions, 0.054

by 0.466 inch; sectional area, 0.0262 square inch; tensile strength, 2768 pounds, or 105,700 pounds per square inch; elongation: in 1 inch, 2 per cent.; in 8 inches, 0.25 per cent.; reduction of area, 8.4 per cent.

Experiment No. 6.

Copper, 59.92; zinc, 40.07; antimony, 0.01 per cent. Melted 5 pounds of copper, added 1.5 ounces of an alloy of 99 copper and 1 antimony, and then 3 pounds, 6.5 ounces of zinc. Rolled from 0.535 to 0.332 inch, a reduction of 38 per cent. No cracks or checks appeared. Annealed and rolled to 0.122 inch, a reduction of 63 per cent. No checks, but a few slight edge-cracks formed. Annealed and rolled to 0.022 inch, a reduction of 82 per cent. Cracked badly on the edges, but no checks appeared. There is apparently little difference in the rolling-qualities between this alloy and brass free from antimony.

The fractures are nearly identical; and this composition, like the pure material, shows no tendency toward crystallization, but a fibrous nature, a condition necessary for a perfect rolling-material.

The fracture of this alloy is shown in Fig. 5.

The following tests were made on a sheet rolled without annealing from 0.122 to 0.048 inch, a reduction of 60 per cent.

Annealed Sheet.—Length, 12 inches; dimensions, 0.0485 by 0.534 inch; sectional area, 0.0259 square inch; tensile strength, 1600 pounds, or 61,800 pounds per square inch; elongation: in 1 inch, 47 per cent.; in 8 inches, 36.3 per cent.; reduction of area, 46 per cent.

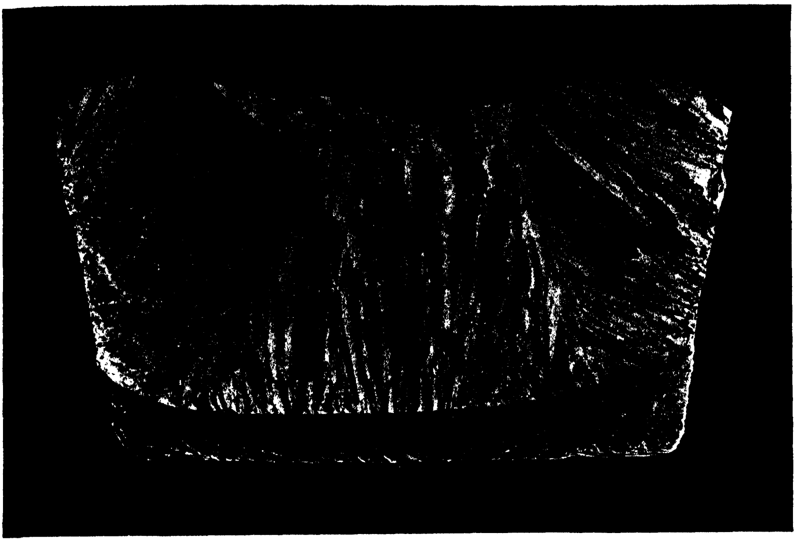
Hard-Rolled Sheet.—Length, 12 inches; dimensions, 0.0485 by 0.551 inch; sectional area, 0.0267 square inch; tensile strength, 2724 pounds, or 102,000 pounds per square inch; elongation: in 1 inch, 10 per cent.; in 8 inches, 2.4 per cent.; reduction of area, 37 per cent.

This alloy forged hot to a thin edge, and bent over on itself and flattened at the end without cracking.

Experiment No. 7.—*A Repetition of No.* 6.

Copper, 59.99; zinc, 40; antimony, 0.01 per cent. Melted 5 pounds, 14.25 ounces of copper, added 1.5 ounces of an alloy

FIG. 1.

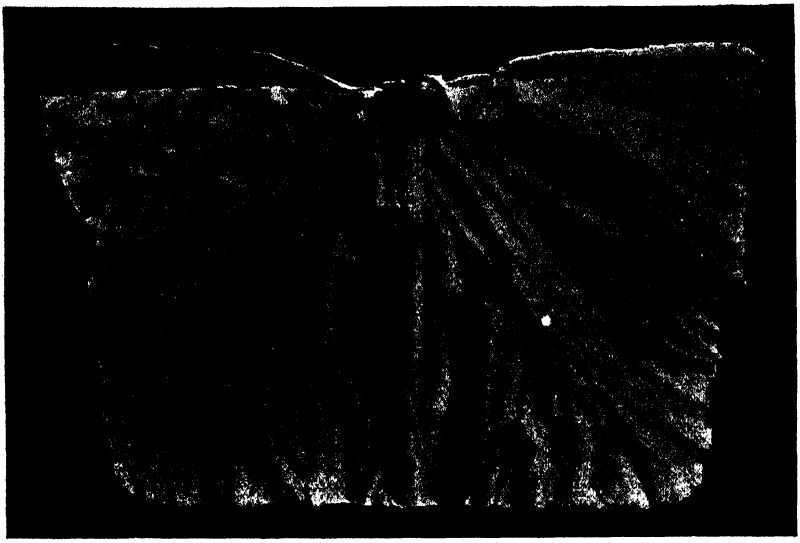


BRASS CONTAINING 0.67 PER CENT OF ANTIMONY.
FRACTURE MAGNIFIED TWO DIAMETERS.

FIG. 2.

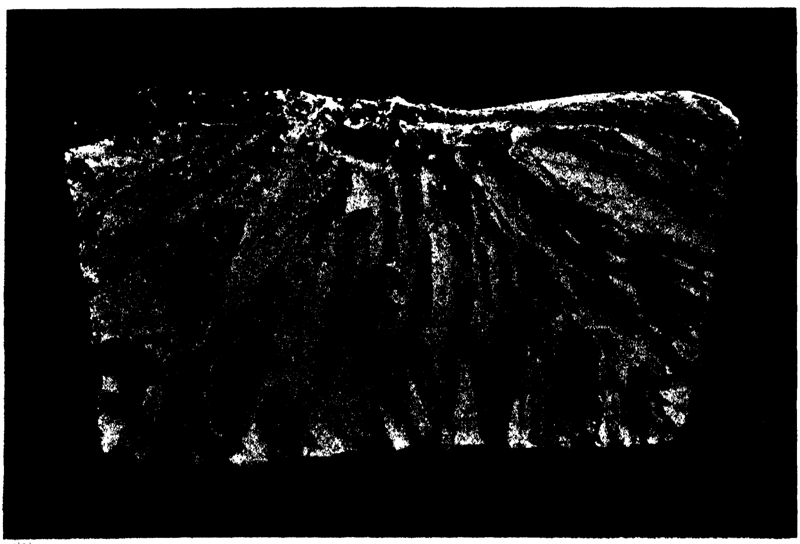
BRASS CONTAINING 0.10 PER CENT OF ANTIMONY.
FRACTURE MAGNIFIED TWO DIAMETERS.

FIG. 3.



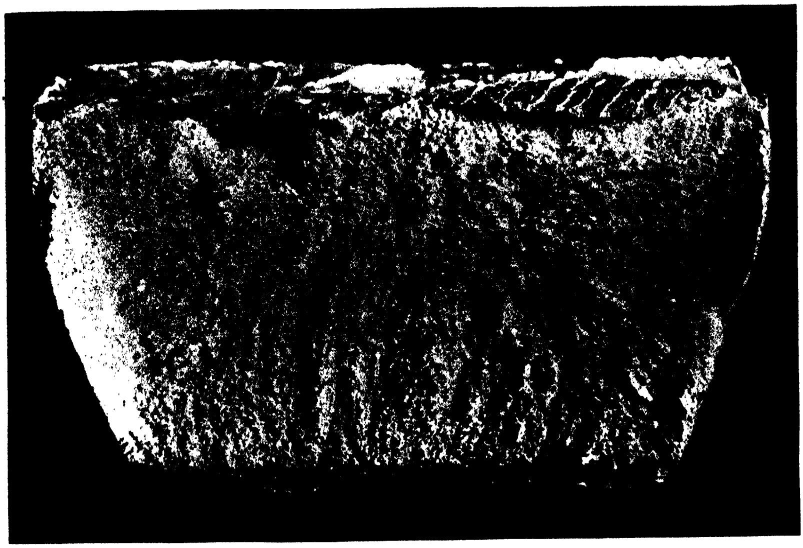
BRASS CONTAINING 0.05 PER CENT OF ANTIMONY.
FRACTURE MAGNIFIED TWO DIAMETERS.

FIG. 4.



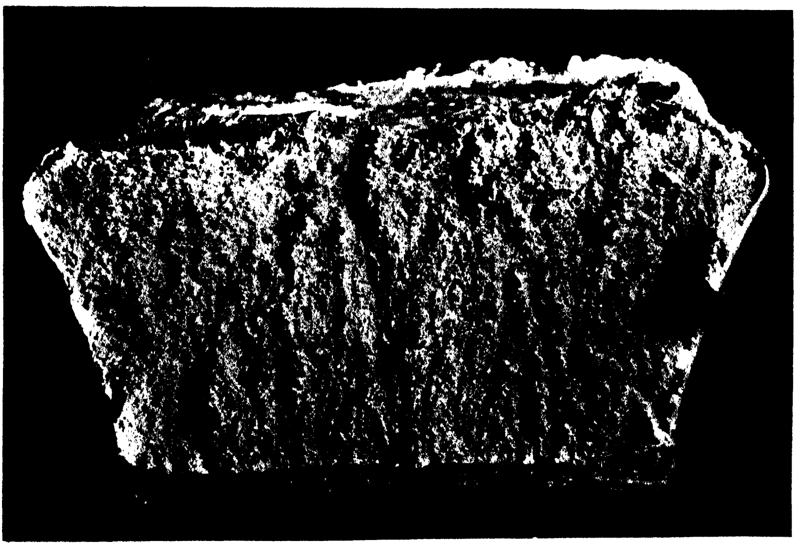
BRASS CONTAINING 0.02 PER CENT OF ANTIMONY.
FRACTURE MAGNIFIED TWO DIAMETERS.

FIG. 5.



BRASS CONTAINING 0.01 PER CENT OF ANTIMONY.
FRACTURE MAGNIFIED TWO DIAMETERS.

FIG. 6.



BRASS FREE FROM ANTIMONY.
FRACTURE MAGNIFIED TWO DIAMETERS.

of 99 copper and 1 antimony, and then 4 pounds of zinc. Rolled from 0.597 to 0.460 inch, a reduction of 22 per cent. No cracks or checks appeared. Annealed, and rolled to 0.370 inch, a reduction of 19 per cent. No checks or cracks formed. Annealed, and rolled to 0.053 inch without annealing, a reduction of 85 per cent. At 0.130 inch, began to crack slightly on the edges; at 0.080 inch, cracked considerably; and at 0.053 inch had cracked badly. No checks appeared at all during the rolling.

The fracture of this alloy was quite tough, and identical with that of Experiment No. 6. (See Fig. 5.)

The following tests were made:

Annealed Sheet.—A. Length, 12 inches; dimensions, 0.053 by 0.468 inch; sectional area, 0.0248 square inch; tensile strength, 1520 pounds, or 61,290 pounds per square inch; elongation: in 1 inch, 52 per cent.; in 8 inches, 33.2 per cent.; reduction of area, 43.7 per cent.

B. Length, 12 inches; dimensions, 0.482 by 0.053 inch; sectional area, 0.0255 square inch; tensile strength, 1546 pounds, or 60,620 pounds per square inch; elongation: in 1 inch, 50 per cent.; in 8 inches, 34 per cent.; reduction of area, 43.8 per cent.

Hard-Rolled Sheet.—Length, 12 inches; dimensions, 0.053 by 0.482 inch; sectional area, 0.0255 square inch; tensile strength, 2780 pounds, or 109,000 pounds per square inch; elongation: in 1 inch, 4 per cent.; in 8 inches, 0.6 per cent.; reduction of area, 16 per cent.

This alloy, like the preceding one (Experiment No. 6) forged and bent upon itself without cracking.

Experiment No. 8.

Copper, 60; zinc, 40 per cent. Melted 6 pounds of copper and added 4 pounds of zinc. Rolled from 0.598 to .500 inch, a reduction of 16 per cent. No cracks or checks appeared. Annealed, and rolled to 0.343 inch, a reduction of 31 per cent. Did not crack or check. Annealed, and rolled to 0.052 inch, a reduction of 85 per cent. No checks appeared, but the sheet cracked badly on the edges, and began to laminate on the ends, proving that the reduction had reached the maximum.

The fracture of this alloy was tough and fibrous, showing no trace of crystallization. (See Fig. 6.)

The following tests were made:

Annealed Sheet.—A. Length, 12 inches; dimensions, 0.055 by 0.474 inch; sectional area, 0.026 square inch; tensile strength, 1526 pounds, or 58,700 pounds per square inch; elongation: in 1 inch, 51 per cent.; in 8 inches, 35 per cent.; reduction of area, 43.9 per cent.

B. Length, 12 inches; dimensions, 0.051 by 0.477 inch; sectional area, 0.0243 square inch; tensile strength, 1492 pounds, or 61,300 pounds per square inch; elongation: in 1 inch, 52 per cent.; in 8 inches, 33.3 per cent.; reduction of area, 43.7 per cent.

Hard-Rolled Sheet.—Length, 12 inches; dimensions, 0.052 by 0 474 inch; sectional area, 0.0246 square inch; tensile strength, 2736 pounds, or 111,000 pounds per square inch; elongation: in 1 inch, 5 per cent.; in 8 inches, 0.75 per cent.; reduction of area, 15 per cent.

This alloy forged hot to a thin edge, and bent over on itself without cracking.

The fact that antimony is an injurious impurity in brass has been known for some time. Kerl* says that experiments carried out in England showed that the presence of as small an amount as 0.001 per cent. of antimony in copper renders it unfit for the manufacture of brass sheet or wire. This figure is abnormal, and the discrepancy between it and that given above as the limit is probably due to the presence in the brass or copper of other impurities, the effect of which was charged to the antimony.

Hiorns† says that antimony is one of the most injurious impurities in brass, but gives no further information.

From the experiments reported above, it is evident that high brass cannot contain more than 0.01 per cent. of antimony and roll satisfactorily; and specifications for copper to meet rigid requirements should limit the contents to that amount, thus giving only about 0.006 per cent. in the brass—a safe amount, providing no other impurities are present. Low brass, on account of its natural softness, is not as susceptible

* *Treatise on Metallurgy* (Translation by Crookes and Roehrig), 1869, vol. ii., p. 82.

† *Mixed Metals*, p. 103.

to impurities as high brass, but the rule given will hold good for brass containing between 60 and 70 per cent. of copper.

In order to find out whether oxygen had any influence on the fracture and rolling-qualities of brass containing antimony, the following experiment was made:

Expcriment No. 9.

Five pounds (0.05 per cent. of antimony) of the alloy used in Experiment No. 3 was melted, and 1 ounce of a 4 per cent. aluminum-bronze was added, introducing enough aluminum to reduce any oxygen present. Poured into the same mould previously used.

The fracture was identical with the alloy containing no aluminum, and the behavior in rolling was the same, proving that oxygen had no influence on the results obtained.

Antimony affects brass in the same manner as phosphorus does steel, producing cold-shortness, and consequently giving a material which is not suitable for rolling. Such cold-shortness is the result of crystallization; and when rupture takes place it is according to the natural cleavage of the crystals. Crystalline metals are not suitable for rolling, a fibrous material giving the best results. Any condition which induces the formation of crystals tends to affect the fitness for rolling. This explains why a brass sand-casting, or plate poured at an excessively high temperature, will not give normal results in rolling.

From an examination of the fractures shown in Figs. 1 to 6, it would seem that such fractures could be used to detect antimony in copper, and to determine the suitability of the metal for the manufacture of brass; since 0.02 per cent. of this element can be detected with certainty. Such a method was used to a considerable extent at one time, and was known as the "spelter-test." It was first brought to the notice of the author at the advent of electrolytic copper in the United States, and was used by him to some extent. Although unreliable to a certain degree, with regard to the nature of the impurity, it can be employed as a quick and safe method to determine whether the brass will roll.

To make such a test, a brass containing 60 per cent. of

TABLE I.—*Summary of Tests of Brass Sheet Containing Antimony.*

Experiment No.	1.	2.	3.	4.	5.		6.		7.			8.		
Copper, per cent Zinc, " Antimony, "	59.46 39 87 .67	60 00 39 90 .10	59 97 39 98 .05	60 06 39 92 .02	62 02 37 96 .02		59 92 40 07 .01		59.99 40.00 .01			60 40		
	Sheet.	Sheet.	Annealed Sheet.	Annealed Sheet.	Annealed Sheet.	Hard Rolled Sheet.	Annealed Sheet.	Hard Rolled Sheet.	Annealed Sheet A.	Annealed Sheet. B.	Hard Rolled Sheet.	Annealed Sheet. A.	Annealed Sheet. B.	Hard Rolled Sheet.
Tensile strength per square inch, pounds	......	....	56,600	58,800	*53,800	105,700	61,800	102,000	61,290	60,620	109,000	58,700	61,300	111,000
Elongation in 1 inch, per cent	...	.....	26	52	53	2	47	10	52	50	4	51	52	5
Elongation in 8 inches, per cent.	......	....	21	34 5	43	0 25	36 3	2.4	33 2	34	0 6	35	33.3	0.75
Reduction of area, per cent	.....	......	27	45	58	8 4	46	37	43.7	43 8	16	43.9	43.7	15
Possible reduction in rolling, per cent	8	25	52	64	76		82		85			85		

* Flaw in metal.

copper is made in the usual manner from the copper in question with pure spelter, and cast in bars 1 inch square and 12 inches long. The bars are fractured and examined. If the fracturing takes place easily, considerable antimony is present; but if with difficulty and the fracture is crystalline, a small amount is present. If the alloy is tough, and fractures with considerable difficulty and no crystallization is noticeable in the fracture, it can be safely assumed that less than 0.01 per cent. of antimony is present, and that the copper will make brass suitable for rolling. Samples of brass containing known amounts of antimony should be fractured and kept for comparison.

Antimony is an inseparable impurity in electrolytic copper, and is partially responsible for its unpopularity. While the best brands of such copper ordinarily do not contain enough antimony to injure brass, oftentimes the content of this element will be so high as to render the process of rolling difficult or uncertain. It is this factor of uncertainty which prevents the universal adoption of electrolytic copper for making brass. Otherwise it would be deemed equal in quality to Lake Superior material.

There seems to be a similarity between the manufacture of basic steel from high-phosphorus ores and the production of copper from ores rich in antimony. In the majority of instances the product is normal; but sooner or later a condition arises which allows the product to become contaminated with the impurity in question, and after a trial by the consumer it is of course condemned. As most copper is used by the consumer without testing, a few experiences of such a nature suffice to hinder any further use of the commodity. The fact that various samples of electrolytic copper contained from 0.001 per cent. to 0.08 per cent. of antimony as the results of many determinations, will serve to illustrate and confirm the above statement.

The results of Experiments Nos. 1 to 8 are collated in the table on the preceding page.

The Manganese-Ore Industry of the Caucasus.

BY FRANK DRAKE, NEW YORK CITY.

(Atlantic City Meeting, February, 1898.)

INTRODUCTORY.

MANGANESE-ORES are known to exist in the Caucasus in a number of localities, viz., in the government of Kutais, near the village of Chiaturi; in the same government near the Choruk river, southward from Batum; and in the governments of Erivan and Tiflis. In smaller quantities they are found in various other places also. The first-mentioned deposits are by far the most important; the others being not only less extensive, but remote from present means of communication, and, from their character, expensive to work, although in some cases the ore is of a superior quality.

All the manganese-ore now known in the market as "Caucasian" ore comes from Chiaturi, no other deposits in the Caucasus having been worked commercially; movements have been made, however, towards the exploitation of some of those on the Choruk river.

THE MANGANESE-ORE DISTRICT OF CHIATURI.

General Remarks, Statistics, Etc.—No other known deposit of manganese-ore can approach that of Chiaturi in capacity for producing large quantities of high-grade ore at a low cost. The deposit is said to have been discovered in 1848, by the geologist Abich, but the first shipments were not made until 1879, when 871 tons* were produced. Since that time, the production has steadily increased, until now the world relies on this deposit for about one-half its supply of manganese-ore. The industry is also of great importance locally, and affords a livelihood to a considerable proportion of the population of the district in which the deposit is situated.

* The term "ton" in this paper signifies the long ton of 2240 pounds avoirdupois.

The total product of the mines of Chiaturi, to date, is estimated at 1,682,400 tons. The following table shows the production and exports of Caucasian ore for a series of years, commencing with 1885, together with the approximate production of the world for the same period. For the year 1898, it is expected that the Caucasian product will be in the neighborhood of 300,000 tons.

Year.	Caucasian Production.	Caucasian Exports.	World's Production.*
1885,	58,722	41,396	140,484
1886,	67,985	53,751	208,289
1887,	51,890	59,523	253,677
1888,	29,401	49,076	186,429
1889,	68,439	55,489	258,935
1890,	168,840	135,492	415,883
1891,	98,670	84,040	323,614
1892,	165,101	129,835	424,746
1893,	166,420	123,228	388,864
1894,	180,533	154,832	403,307
1895,	160,277	171,608	
1896,		193,641	
1897,	231,868	201,612	

Location and Topography of the District.—Chiaturi is a village of the district of Sharopan, which forms part of the Trans-Caucasian Russian province of Kutais. The village lies on the Kvrilli river (a tributary of the Rion, which enters the Black Sea near Poti), and is now connected by a narrow-gauge railway with the station Sharopan (25 miles to the southwest) on the main line of the Trans-Caucasian railway. The position of the village, and its railway connections with other parts of Trans-Caucasia are shown by the accompanying map, Fig. 1.

Topographically, the vicinity of Chiaturi is characterized by high, and in some instances precipitous, mountain spurs left by the erosion of the valley of the Kvrilli river, and of the ravines through which flow its tributary rivulets.

The Ore-Deposit.—The ore of Chiaturi occurs in a bedded deposit, lying almost horizontally, near the tops of the lofty hills in the vicinity of the village, and at an altitude of 1000 feet above the Kvrilli river. The action of the elements in forming the rugged topography observed in this region has carried away, perhaps, more than one-half the original deposit.

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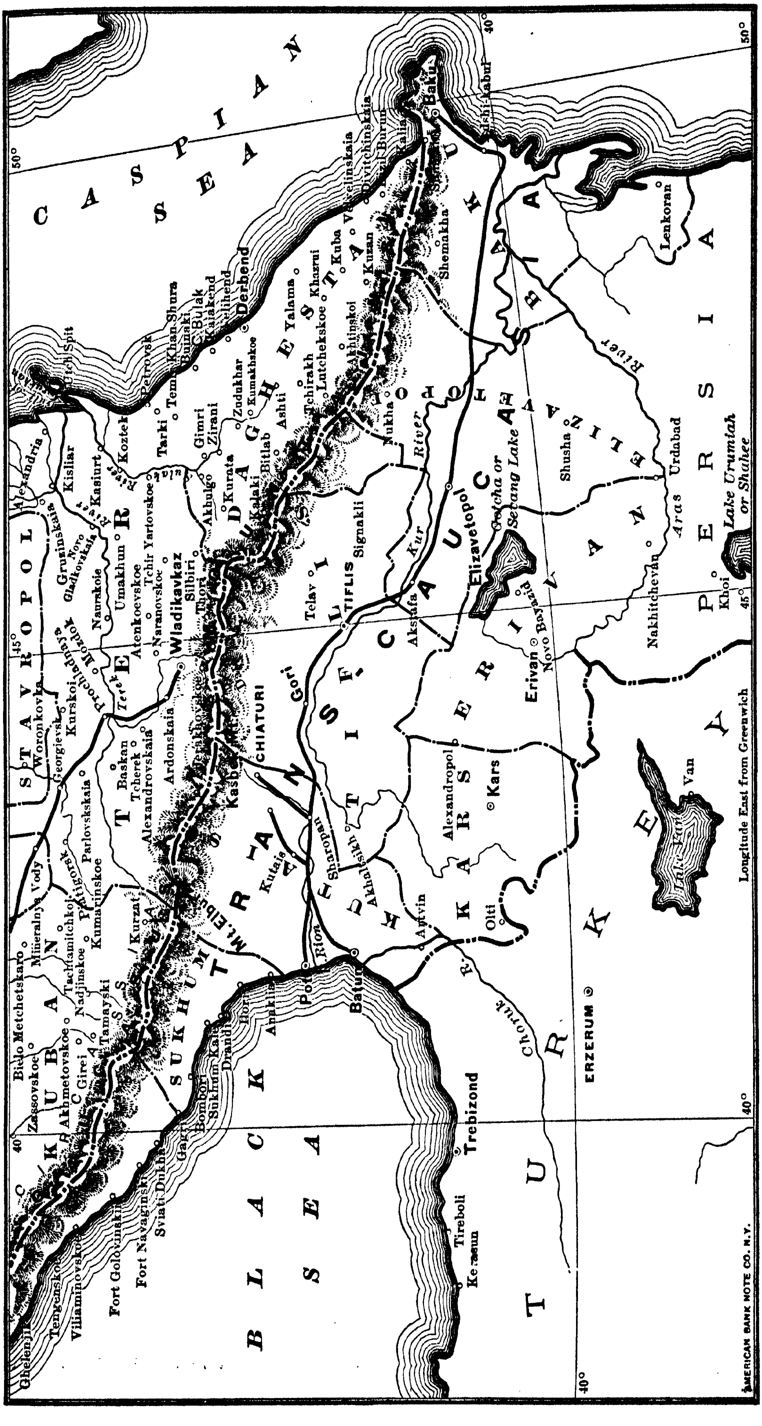
* Compiled from the *Mineral Industry*, vol. vi., and the *Reports of the U. S. Geological Survey.*

FIG. 1.



The Caucasian and Trans-Caucasian Provinces of Russia.

The existing bed has been opened on seven of the mountains near Chiaturi. Of these, three (designated locally as Perivissi, Chocrotti and Itvissi respectively) lie to the south of the Kvrilli river, and four (Organyi, Zedorganyi, Gwimavi and Darkvetti) lie on the north side of the river.

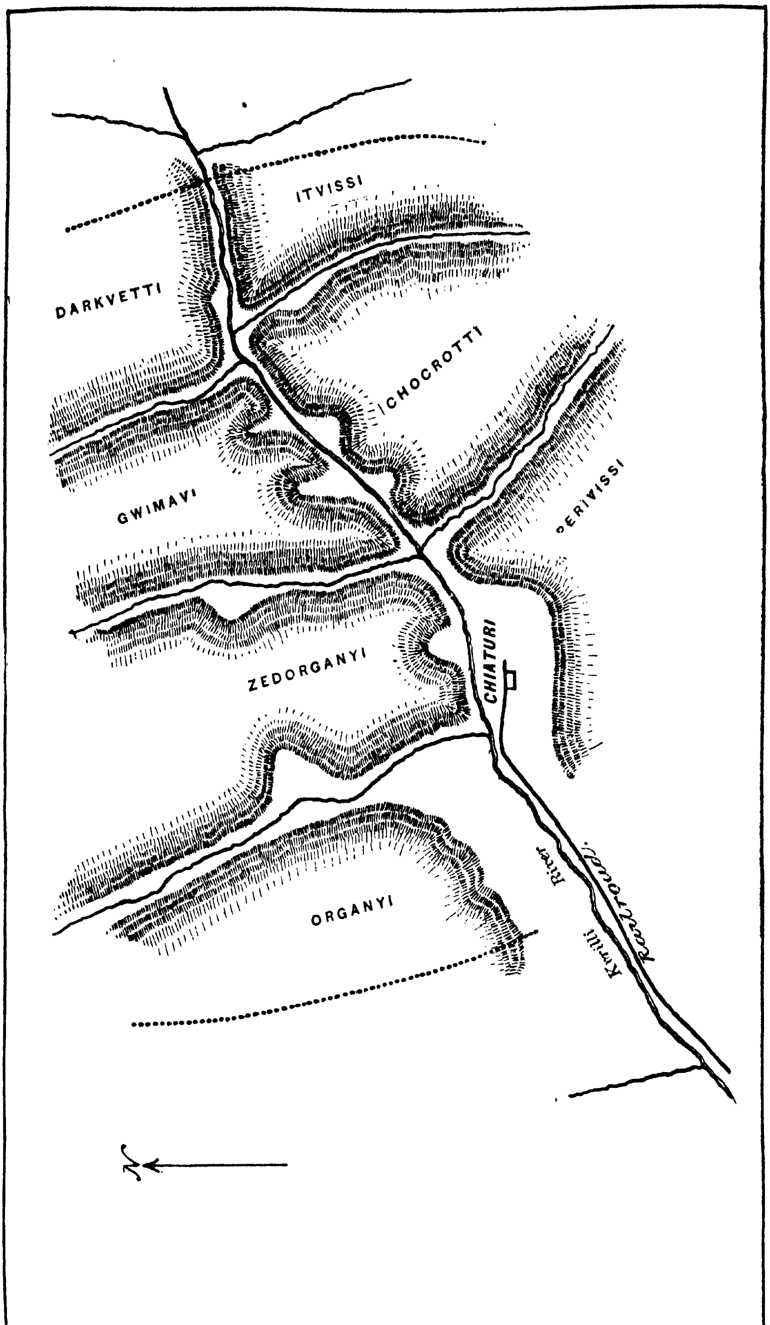
Fig. 2 is a sketch-map of the vicinity of Chiaturi, showing the position of the seven mountains named, and upon which the dotted lines indicate, approximately, the limits beyond which it is supposed (from the results of such explorations as have been made) that the ore-bed becomes too poor to repay working. Fig. 3 is a diagrammatic section across the valley of the Kvrilli river, showing the ore-bed lying near the tops of the mountains. Fig. 4 is from a photograph taken near the top of the mountain called Zedorganyi, and gives a good idea of the manner in which the deposit lies. Fig. 5 is a partial view of Chiaturi, looking down the Kvrilli river from Zedorganyi. In the foreground on the left are shown the ore-storage grounds.

The bed occurs in a brown sandstone of Miocene age, and has an average thickness of between 6 and 7 feet. Its dip, which is slight, and fairly regular, is southeasterly. Slight faulting of the formation has occurred in some instances; but few folds are observed, and the bed is free from sudden or extreme variations from the average thickness.

The deposit has a distinctly stratified structure and is composed largely of pyrolusite, but other oxides of manganese also occur. In many instances, strata of sandstone, or of loose, friable arenaceous and calcareous material, are intercalated with the manganese-ore; such strata vary in thickness from a small fraction of an inch to as much (in some of the intercalated layers of sandstone) as 10 inches, or a foot.

As no surveys of the ore-fields at Chiaturi have ever been made, it is impossible to estimate, with any approach to accuracy, the amount of ore existing there. The area given by the government engineers as embracing the whole of the bed at present known is 126 square versts, or about 55 square miles; but this includes not only the existing bed, but also the unknown area of the valleys and ravines intervening between its various segments. It is quite certain, however, that an area of more than 22 square miles of the present surface is underlain by ore available for mining; and on this basis it is estimated

Fig. 2.



Sketch-Map of the Vicinity of Chiaturi.

that, even if the crude and wasteful methods now pursued should be always continued, the bed will yield more than 80,000,000 tons of marketable ore.

Chemical and Physical Characteristics of the Ore.—The ore of Chiaturi is high in manganese. In some localities, over limited areas, the material composing the bed, without sorting or cleaning, carries 50 per cent. of the metal, and the average for such material is probably from 40 to 45 per cent.; while ore from which barren material has been thoroughly separated contains, in some instances, as much as 61 per cent. The imperfectly sorted ore that is exported generally runs from 46 to 56 per cent. in manganese, the average, probably, being between 51 and 52 per cent. With proper care, no ore from Chiaturi should carry less than 51 per cent.

Phosphorus, in the ore as shipped at present, averages about 0.16 per cent., and silica not above 8 per cent.

All of the figures given above refer to ore dried at 212° Fahrenheit.

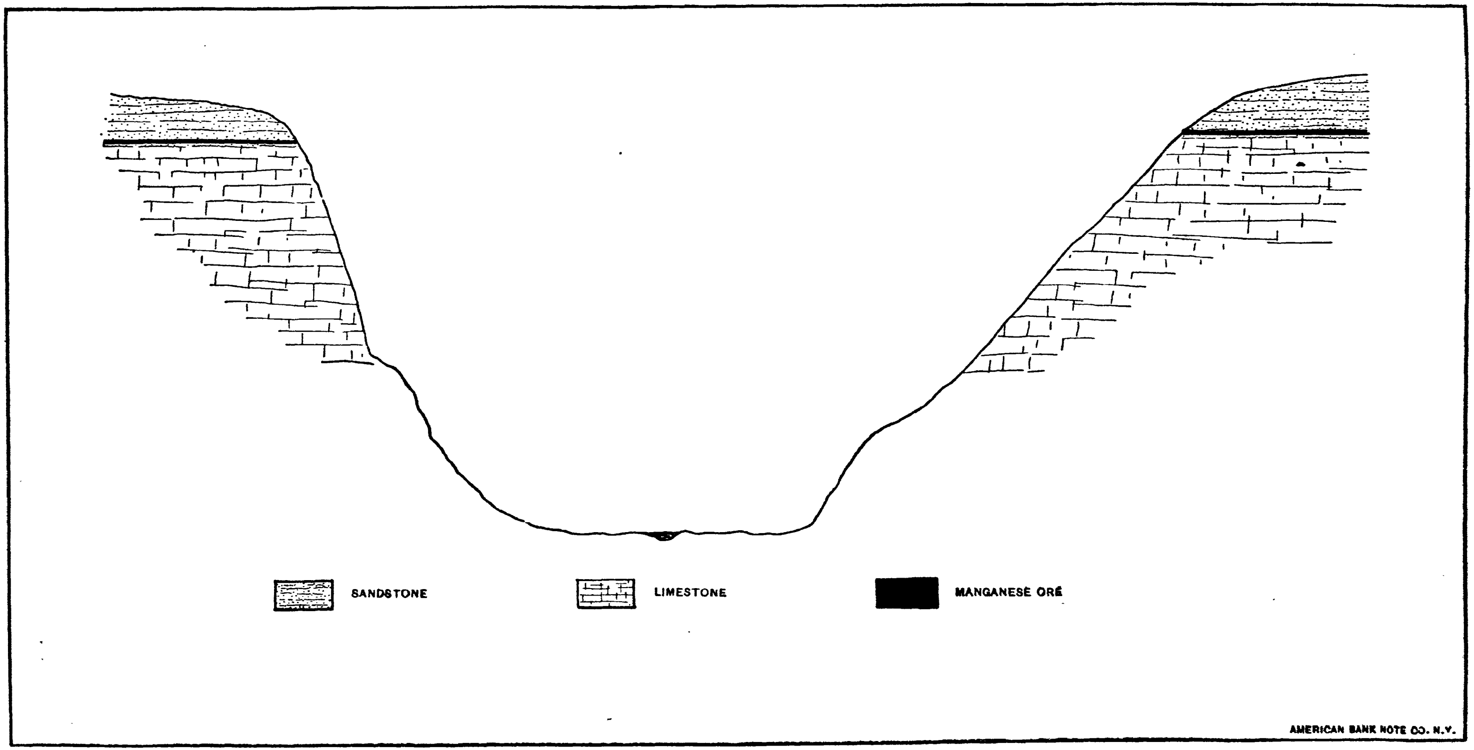
The following is a complete analysis of a sample of very well sorted and cleaned ore from Chiaturi:

Ore dried at 212° F.	Per cent.
Manganese peroxide,	86.25
" protoxide,	0.47
Iron peroxide,	0.61
Oxide of copper,	0.01
" nickel,	0.30
Alumina,	1.74
Lime,	1.73
Magnesia,	0.20
Baryta,	1.54
Potash and soda,	0.22
Silica,	3.85
Carbonic acid,	0.63
Sulphur,	0.23
Phosphoric acid (0.141 P),	0.323
Combined water,	1.850
	99.953

Metallic manganese, 54.90 per cent.

The physical characteristics of the manganese ore of Chiaturi are unfavorable in that the proportion of large pieces obtained is small, and much of the ore is quite soft, and grinds to a fine powder during the handling incidental to mining, clean-

Fig. 3.



Diagrammatic Section Across the Kvrilli Valley.

ing and transportation. The loss of ore during transport is thus larger than it otherwise would be, while the high proportion of fine ore is sometimes considered objectionable by consumers.

Ownership of the Deposit.—The ore-fields at Chiaturi are owned by an unknown, but very large, number of proprietors, mostly natives, who hold parcels of ground varying in size from half an acre to 40 or 50, or possibly more, acres. Some of these proprietors mine the ore from their holdings themselves, while others lease the mining right, receiving therefor a royalty varying from 10 to 50 cents per ton. In general, owners of the surface-soil own also the underlying mineral; but in some instances peasants, and others, have purchased of the landlords surface-rights, which do not include the right to the ore underneath.

Labor and Wages.—The labor market at Chiaturi is recruited from the peasantry of the district of Sharopan. These people, unable to earn more than a scanty subsistence from their unproductive lands, and consequently always more or less poverty-stricken, are glad to accept, for a portion of each year at least, even the low wages offered at Chiaturi for mining, cleaning and transporting ore.

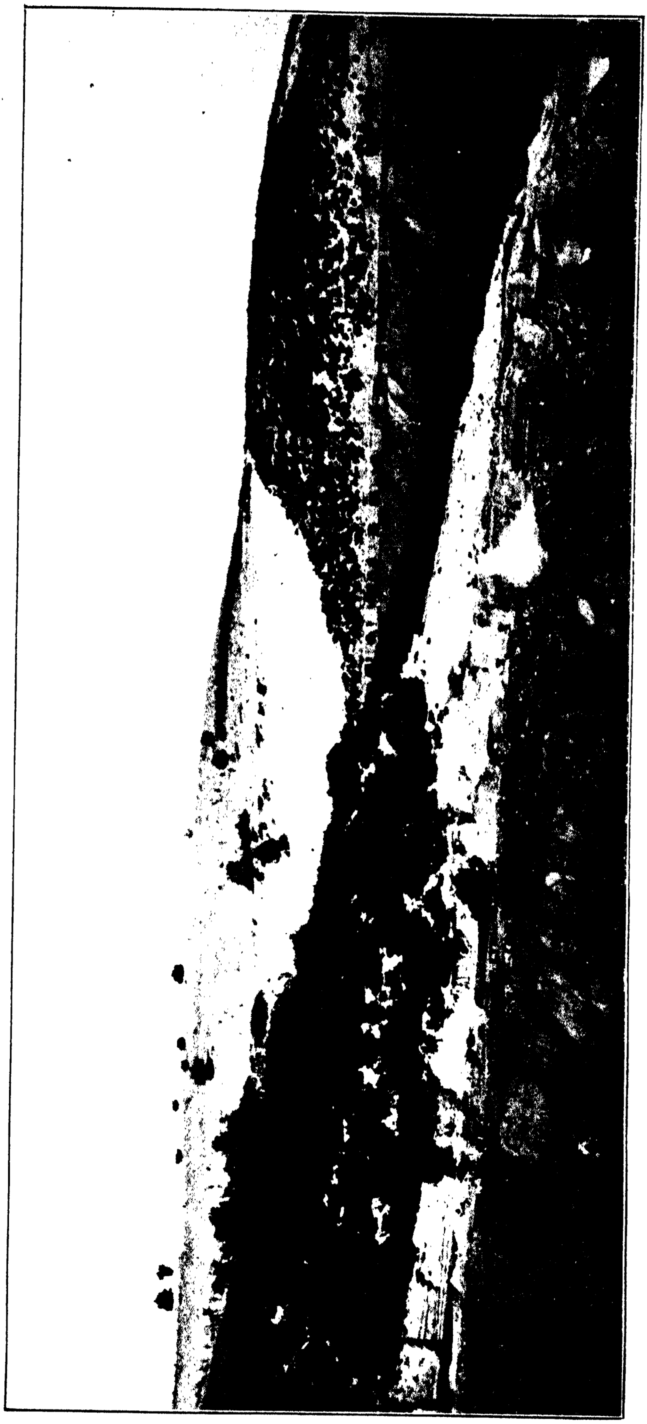
The fact that there is no regular mining population in the district, and that the men obtainable for operating the mines have, as a rule, some agricultural land of their own, and hence, apart from the production and transportation of ore, an occupation to which they return with more or less frequency, introduces an element of uncertainty into the labor-supply which is a disadvantage to the industry.

The wages earned at Chiaturi are about the same for all classes of labor, and average probably not more than 40 cents per day for a man, and about 70 cents for a man and horse. But, small as these wages are, they seem to meet the meager requirements of the workers.

The working day begins at sunrise and ends at sunset, with intermissions of various lengths for meals and rest. The actual hours of labor probably do not average, the year round, more than 8 per day.

The average number of men employed in the ore-industry at Chiaturi is estimated at about 2700.

Fig. 4.



Chiaturi Manganese Ore-Deposit, near the top of Zedorganyi Mountain.

Fig. 5.

Chiaturi. Ore-Storage Grounds in the Left Foreground.

METHODS AND COST OF MINING AT CHIATURI.

Characteristics of the Deposit as Regards Facility of Working.—Seldom can underground mining be undertaken under natural conditions so favorable as those existing at Chiaturi. Not only is the bed of a sufficient thickness for economically breaking the ore, but the enclosing rock is so firm that very little timbering is required, and the ore itself is mostly so soft that it can be advantageously mined without the use of explosives. Moreover, there is no water to contend with, nor are expensive shafts or other openings necessary in order to reach and mine the ore.

The characteristics of the crude ore are also favorable for inexpensive beneficiation by hand-sorting, as now practiced, or by washing. The distinct bedding-planes which characterize the material of the deposit, and the weakness of adhesion along these planes, result in the separation, in large part, of the pieces of ore and gangue during the process of mining; and when this does not occur, this separation afterwards, by hand, is rendered easy by the same characteristics.

The features mentioned above, and the additional circumstance that there is a considerable difference in specific gravity between ore and gangue, indicate that excellent results would be obtained by washing the ore mechanically.

Mining.—The native miners have adopted the obvious method of attacking this deposit where it is exposed, or may be easily uncovered, on the mountain-sides. Their practice is to drift on the bed, perpendicularly to the exposed face, until the drift becomes so long that the expense of removing the ore from it in wheelbarrows or baskets is too great to permit further progress. Several such drifts may be commenced at one time, separated by distances determined by the fancy of the operator, and extending in directions more or less approaching parallelism. Later, cross-drifts may be driven, leaving the roof supported by pillars, which may afterwards be "robbed" until the roof falls in; or, as is frequently the case, no cross-drifts are driven, but the main drifts, or parts of them, are enlarged until they become extensive stopes, of irregular size and shape, and cave in, and are abandoned. By these methods probably more than one-half the ore in the ground worked at Chiaturi is being

lost. Most of this loss might be avoided by the adoption of a rational system of mining.

The work of breaking ore at Chiaturi is extremely easy. As already observed, the greater part of the ore may be readily mined with a pick, and the floor and roof are so firm as to give no trouble. It is largely the practice to mine out the whole thickness of the bed, sorting the ore outside; but in instances where thick strata of sandstone, or of ore too poor, from admixture of gangue, to be profitably hand-sorted, occur, it is common to break such waste material separately, and to stow it underground. This stowing is seldom or never done systematically, the waste material being usually disposed of in any manner convenient for the worker at the time. The transportation of ore from the working-face to the cleaning-ground is accomplished almost entirely by the use of baskets and wheelbarrows. But in a few instances, in the longer drifts, a track of wooden rails has been laid, and a primitive mine-car has been placed in operation. Timbering is but slightly employed in the mines, and under any appropriate system of working would not, to any extent, be needed.

The workings described generally do not extend more than 300 feet back from the face of the mountain; but, in a few instances, they have been carried to a horizontal depth of from 400 to 500 feet. Thus only the outer edges of the ore-bed are being worked.

Ore-Cleaning.—The crude ore broken underground is not in condition for shipment, containing, as it does, considerable intermixed sandstone or loose sandy and calcareous material, which must be separated. The method at present employed at Chiaturi for this purpose is the following: The ore brought from the mine is dumped on the level ground at the mouth of the tunnel, and (either immediately or after it has been spread out and allowed to dry for some time) a laborer with a shovel throws it against a wire screen of about ½-inch mesh, which is fixed in an inclined position, one end resting on the ground, and the other supported by a post or prop. The material passing through the screen is considered as waste, and undergoes no further treatment. The portion failing to pass through the screen falls in front of it, and is hand-sorted, pieces of bar-iron and similar implements being used, when necessary, to separate

adhering portions of waste-material from the pieces of ore. The proportion of clean ore yielded by the crude ore of Chiaturi under this treatment varies considerably in different localities, but it is thought that it averages about one-third. It has been stated with reference to some localities that a yield of 50 per cent. of clean ore is obtained; on the other hand, there are localities where the yield would not exceed 20 per cent.

The Possibilities of Mechanical Methods for Treating the Crude Ore.—The general crudeness and inefficiency of the hand-sorting and cleaning process described above have led to many suggestions for its improvement or replacement. Of these the two following may be noticed:

1. The substitution of mechanical screening for hand-screening in the present process, together with the adoption of mechanical means (such as picking-tables or belts) for facilitating the sorting, cleaning and handling of the ore. Such a change would no doubt result in some reduction in the cost of the cleaning-process, or, for the same cost, it would permit either an improvement in the grade of the product or an increase in the proportion of clean ore yielded by the crude. Inasmuch, however, as the cost of the present process is quite low, no great betterment of existing conditions could be accomplished in this way.

2. Mechanical washing naturally suggests itself as a suitable method for the treatment of the crude ore. So far as known, but one attempt at experiment in this direction has been made, the results of which were highly encouraging. These experiments indicated that by this means a general product carrying from 59 to 60 per cent. of manganese may be obtained from the ore of some localities, and also that, owing to the possibility of saving much ore that now goes to waste, a proportion of clean ore would be yielded by the crude ore much larger than at present. The cost of washing would be considerably greater than that of the present hand-cleaning process, but this would be more than counterbalanced by the increased quantity and value of the product.

The Cost of Producing Ore.—Mining and cleaning are generally let together, by contract, at a fixed rate per "cubic sagene" of clean ore, the contractors mining the ore, transporting it outside and cleaning it, and the operator or owner supplying

the timber, lights, etc. The Russian standard cubic sagene contains about 343 cubic feet, and this volume of clean broken ore would weigh about 23 tons; but at Chiaturi "cubic sagenes" of from 1½ to 2 times this capacity are used, so that the weight of a cubic sagene of ore varies from 34 to 46 tons. Contracts are taken by parties composed of two men or more, and the rate paid per cubic sagene, of clean ore, is from $10 to $25, according to the distance of underground tramming, proportion of waste contained in the crude ore, etc. The number of days required for one man to mine and clean a cubic sagene varies from 20 to 80.

Under these widely differing conditions it is difficult to arrive at the average cost for the labor of mining, tramming and cleaning; but for the more favorably conditioned properties it is thought that the average is about 39 cents per ton of clean ore, and that the following estimate of the total cost at the mine per ton of clean ore is a fair approximation:

Labor for mining, tramming and cleaning,	$0.39
Timber and other supplies,	0.04
General expense,	0.16
Royalty,	0.17
Subscription to Producers' Association,	0.16
	$0.92

In explanation of the last item (which is really a part of the general expense of production and transportation from the mines to the Chiaturi railway station) it may be said that the manganese-ore producers all contribute to a fund which is expended, under the supervision of officials of the government, for objects of general utility, such as police service, building and maintaining roads, etc.

Transportation to the Seaboard.

The transportation of ore from the Chiaturi mines to Poti, on the Black Sea (which is the principal shipping port, although a small quantity is shipped each year from Batum) is effected in three stages; the ore being first brought down the mountains from the mines to the railway station at Chiaturi, then carried on the narrow-gauge railroad to Sharopan, and from that point taken over the main line of the Trans-Caucasian railway to Poti.

Transportation from Mines to Chiaturi.—The position of the ore-bed near the tops of the mountains in the neighborhood of Chiaturi has already been described. The distance from the village, of the workings at present opened, varies from 1 to 4 or more miles, and transportation from the mines is, in large part, effected over narrow trails which traverse in zigzags the precipices and steep slopes of the mountains. Some of these trails are impassable, except for the most sure-footed pack-animals, while over others it is possible to use the primitive two-wheeled ox-cart, known in Georgia and Imeretia as an "arba."

The animals, as well as the carts, employed for this work are the property of the peasants engaged in it; and the latter are paid for the transportation of ore (including loading and unloading) at rates varying, in good weather, from 65 cents to $1.30 per ton, according to the distance from the mine to Chiaturi. In wet weather the steep mountain-trails are dangerous and almost impassable, and the cost of transportation over them becomes abnormally high; at the same time the quantity it is possible to transport is greatly decreased.

The adoption of tramways for transportation from the mines to the village is perfectly feasible, and would not only reduce the cost very materially, but would render it, as well as the quantity transported, independent of the varying condition of the mountain roads.

Transportation from Chiaturi to Sharopan.—Toward the end of 1893, the narrow-gauge railway connecting Chiaturi with Sharopan was completed. Before that time ore was transported from Chiaturi to the Trans-Caucasian railway (at Kvrilli, a station near Sharopan) by means of carts and pack-animals. Now the ore received at the station is stored near the railway-tracks, and is afterwards loaded on the narrow-gauge cars. There are no special facilities for loading, the work being accomplished by carrying the ore on the cars in baskets. The cars hold about 4 tons each. They are of the simplest type, and are not provided with drop-bottoms, or other special arrangement for facilitating unloading. On arrival at Sharopan, the ore-train is run out on a trestle, extending over a platform which is level with the floors of the cars of the broad-gauge road to Poti. The narrow-gauge cars are unloaded by the

use of shovels, and the ore, falling on the platform below, is transferred by shovelling and the use of baskets, to the broad-gauge cars.

The freight-charge made by the Government Railway Administration, for transporting ore from Chiaturi to Sharopan, a distance of 25 miles, is $3.28 per ton, or at the somewhat high rate of 13.1 cents per ton-mile. To this must be added 36 cents per ton for station-expenses, weighing, loading at Chiaturi and transferring at Sharopan, making the total expense of taking ore from storage at Chiaturi, and placing it on board the broad-gauge cars, $3.64 per ton.

Assuming that a freight-rate of 40 cents per ton, for transportation from Chiaturi to Sharopan, would remunerate the capital invested in the railway, it is evident that, should a formidable competitor to the Chiaturi deposits ever arise, the Russian Railway Administration would have it in its power to immediately reduce the cost at which the ore may be delivered in foreign ports by $2.88 per ton; and it could effect a still further reduction by making the Chiaturi branch of standard gauge.

Transportation from Sharopan to Poti.—The distance from Sharopan to the Poti railway-station is 82 miles, and from the station to the pier, where ore is loaded for export, 1⅓ miles. Ore is carried over the Trans-Caucasian railway in covered and in open box-cars, both types holding about 10 tons. Like the cars used on the narrow-gauge road, they are not provided with any arrangements for facilitating unloading, which has to be done at Poti as at Sharopan, with shovels and baskets.

Ore is frequently discharged at Poti station and kept in store there for some time (there being no room for storage on the pier) and then reloaded and taken to the pier, from which it is at once carried in baskets on board vessels waiting to receive it.

The freight from Sharopan to Poti station is 52 cents per ton, to which must be added an amount averaging about 29 cents per ton for station-expenses, storage, transfer from station to pier, etc., making the total cost of transferring ore, loaded on cars at Sharopan, and placing it f. o. b. vessel at Poti, 81 cents per ton.

THE EXPORT AND SALE OF CAUCASIAN ORE.

Great Britain is at present the largest consumer of Caucasian ore, taking about 100,000 tons annually. Then follow Germany,

France, the United States and European Russia. The ore is distributed among consumers through the medium of brokers and merchants in London and other centers, who either sell it on commission or else buy it from exporters in Poti, who, in turn, have connections among the producers at Chiaturi. In some instances, however, exporters are also engaged in mining ore; and some of the European merchants who handle manganese-ore have representatives at Poti or Chiaturi, and buy directly from the producers.

The ocean-freight from Poti to the ports of Western Europe varies from 9 shillings per ton to 15 shillings, or even more, and is greatly influenced by the demand for grain-tonnage. At present, the freight to English ports is about 12½ shillings, equivalent to $3.03 per ton, to which must be added 12½ cents for insurance, sampling, checking weights, etc. Freights to America are generally about 35 cents per ton higher than those to England.

A duty of 16½ cents per ton is levied on exports of manganese-ore from Poti, the proceeds being applied to the maintenance and improvement of the port.

CONCLUSION.

The Total Cost of Caucasian Ore.—The cost (neglecting moisture and loss in transit) of Caucasian ore, delivered c. i. f. at English ports, may be placed about as follows:

	Per long ton.
Cost of production,	$0.92
Transportation to Chiaturi,	.71
	1.63
Loading and storage at Chiaturi and transferring at Sharopan,	.36
Freight from Chiaturi to Sharopan,	3.28
" " Sharopan to Poti Station,	.52
Unloading, storage, etc., at Poti,	.29
Port-duty,	.16½
	$6.24½
Ocean-freight, insurance, sampling, etc.,	3.15½
	$9.40

The Present Commercial Condition of the Industry.—The price in England of Caucasian ore, delivered as above, is at present about 9½ pence per unit of metal, the percentage of the latter

being determined on the basis of ore dried at 212° F., and a deduction being made from the gross weight for the moisture as shown by sample. At this rate a 52 per cent. ore is worth $9.97 per ton, dry weight.

While prices, ocean-freights and the manganese-percentages in the ores vary considerably, and the figures given above for cost of production and for some other items are only approximate, it is evident, considering that moisture and loss in transit amount to from 5 to 7 per cent., or even more, that there is little or no profit in the business. This, in fact, has been, for several years past, the complaint of all engaged in it.

The reasons for this condition are to be found in the fact that, although the deposit at Chiaturi is practically free from foreign competition, the local conditions affecting it (particularly the enormous quantity of ore it contains, the large number of proprietors among whom it is distributed, the ease with which the production of ore may be entered upon, and the lack of other sufficient occupation for the inhabitants of the region), are such as to promote over-production and severe competition among the producers of the ore, which keep prices at a level affording to those engaged in the industry only a meager subsistence.

The writer desires to acknowledge his indebtedness for information and assistance to Messrs. Hassfeld and Saet, ore-factors, of Kutais and Poti; and to Mr. Citelin, District Engineer at Kutais. Information has also been obtained from the *Report on the Manganese-Ore Industry of Sharopan*, written by Mr. P. Stevens, British Consul at Batum, and published by the British Foreign Office.

Emery, Chrome-Ore and Other Minerals in the Villayet of Aidin, Asia Minor.

BY W. F. A. THOMAE, A.R.S.M., LONDON, ENGLAND.

(Atlantic City Meeting, February, 1898.)

THE Villayet of Aidin is a province in Asia Minor which has a coast line extending from opposite the island of Mitylene to beyond Makri, opposite the island of Rhodes, and embraces almost the entire basins of the two principal rivers, the Sarabat

(Hermus) and the Mender (Mæander), besides some smaller ones. The principal town is Smyrna, the center of trade of the district, from which two railways run into the interior along the valleys of the two rivers just mentioned.

It is not proposed to give here an exhaustive account of all mineral occurrences in this district, but only of such as the writer has personally examined. The former would require years of investigation, as the area is vast, and little or nothing is yet known of a great part of it.

Topography and General Features.

The topography of the country is very characteristic. Hills and mountains rise everywhere with more or less jagged outlines, bounding wide, fertile plains, often as flat as a billiard-table. The plains are cultivated with barley and other cereals, or form huge vineyards and orchards, chiefly of fig and olive, also of orange, pomegranate and other fruit-trees. The hills and mountains are barren and uncultivated, mostly covered with low, prickly shrubs, sometimes with pine-trees, and occasionally with oak; the timber being, with a few exceptions in the interior, scattered and scanty. The creeks and water-courses, intersecting the mountains on their way to the plains, often water perfect gardens of luxurious growth, including fig- and olive-trees, sycamores, poplars, lime-trees, willows, etc. In other places they rush or trickle over huge boulders in a stony ravine, where a few stunted shrubs form the only vegetation.

The mountains rise to over 7000 feet, heights of from 3000 to 4000 feet being quite numerous. The general level of the country rises considerably towards the interior, till near the source of the Mender the lowest valleys are near 3000 feet above the sea.

Geology.

By far the greater part of this country is composed of limestones and schists, and presents a fine example of orthodox regional metamorphism. The shell-, mud- and other beds, originally deposited over a sea-bottom, extending probably far beyond the region here described, have been completely metamorphosed, the limestone to pure white saccharoid marble, now

covering large areas, and the other beds, interstratified with it, to schists of various kinds—mica, chloritic and hydro-mica, often changing gradually the one into the other, and sometimes passing insensibly into gneiss. In several localities the schists contain regular octahedra of magnetite up to half an inch in diameter.

In one place on the coast, a little below the island of Scios, pure red clay-slates occur, carrying thin beds of pyrolusite.

The general strike of these formations throughout the country is about E. and W., though locally the schists are much folded, and strike and dip in all directions. The average dip is steep but not so uniform, and is not always apparent. South of Aidin it is generally S., whilst north of Aidin it appears to be to the N. Further north, again, at Odemish, the dip is S., indicating several parallel foldings of the strata, the number and extent of which observations were not sufficient to determine.

In several places serpentine belts occur. These appear to be interstratified with the marbles and schists, and would thus point to a result of the general metamorphosis on original possibly glauconitic deposits; but further investigation is necessary before it can be definitely asserted that they are not alteration-products of intrusive sheets of basic olivine rocks.

Round the bay of Smyrna, extensive areas consist of volcanic lavas and tuffs, chiefly trachyte.

Overlying the metamorphosed formations there are found in places, such as between Buladan and Ala Shehr, and south of Chesmé on the coast, undisturbed Tertiary beds of sandstone and soft chalky limestone, lying flat or dipping at a very slight angle, and full of small fossil shells, chiefly gasteropods.

Still more recent deposits are evidenced by the huge fan-talus brought down from the mountains north of Aidin, and on the slopes of which the town is very picturesquely situated.

Mineral Deposits.

Of the formations here described the crystalline limestone, which was largely quarried by the ancients for building-purposes, is the home of emery and silver-lead. The serpentine contains chrome-ore and manganese-deposits; the schists, de-

posits of magnetite and hematite iron-ore, mispickel, more or less auriferous, and antimony; the trachyte, veins of quartz carrying galena, blende, antimonite, copper and iron pyrites, and gold and silver in small quantities.

1. *Emery.*

First in the ranks of mineral occurrences stands emery. This is essentially a mixture of corundum and magnetite, and occurs in irregular pockets in the marble in numerous localities.

Its existence was first investigated in 1849 by Dr. J. Lawrence Smith, who published a paper on his researches in 1850.* It was not till nearly twenty years later that much attention was directed to the exploiting of these Asia Minor deposits in competition with those of the island of Naxos, which held the practical monopoly of the emery trade. Then Messrs. Jackson, Charnaud, and later Mrs. Abbott, largely developed the business, and now there are several producers.

Reference to the accompanying map, Fig. 1, will show the locations of emery-deposits that have been and are now being worked. It will be seen that the locations are most numerous in the belts of limestone near Tireh and the Gumush Dagh. Many of these deposits are now exhausted, at least so far as profitable working is concerned, though there are others that have not yet been touched. The chief supplies come at present from some of those on the Gumush Dagh, and some further south on the slopes of Ak Sivri.

The deposits worked are of two kinds: (1) the mineral solid and *in situ*, when it is known as "rock-emery;" and (2) emery detritus, resulting from the weathering of the former.

(1) *Rock-Emery.*—These deposits, as already mentioned, are irregular pockets in the limestone. Their maximum width varies from a few feet up to 200 feet, the length also varying up to about 300 feet, and the depth from 10 to 50 feet, as shown in the workings; but the deposits have a greater depth in cases where there has been, so far, enough ore near the surface to make deeper sinking, with its attendant extra cost, unnecessary. The time will no doubt come when the shal-

* "Memoir on Emery," *Am. Jour. of Science*, 2d series, vols. x., 1850, p. 354, and xi., 1851, p. 53.

lower and more easily accessible deposits will be exhausted, and then deposits now abandoned as unprofitable will again be worked to greater depths. But this time is yet distant.

Sometimes the deposits have an elongated form, a width of 5 to 6 feet with a length of 200 to 300 feet, and then their outcrop, rising boldly over the softer limestone, has the appearance of a reef or fissure-vein.

The walls of the deposits are often exceedingly irregular, the limestone intruding and receding most unexpectedly. The demarcation between it and the deposit is, however, beyond the range of decomposition, always distinct, and there is no gradual merging of the one into the other, though the limestone in juxtaposition is often stained and veined with brown seams. In a few instances, where the decomposed portions of deposits seem to merge into the limestone, this is apparently the result of secondary alteration, subsequent to the formation of the deposits.

The ore forming the deposit is a solid mass of emery, contaminated, however, with an admixture of various silicate minerals, chief among which are margarite ($CaO,2Al_2O_3,2SiO_2,H_2O$); biotite ($Al_2O_3,SiO_2 + 3RO,2SiO_2$); chlorite ($2[2RO,SiO_2] + Al_2O_3,3H_2O$); and chloritoid ($[FeMg]O,Al_2O_3,SiO_2,H_2O$).

These are usually present as an intimate mixture with the emery, and, according to their quantity, operate to deteriorate the quality of the latter, or make it useless. Often, however, they also occur in distinct veins, from a fraction of an inch to about 2 inches thick, traversing the ore irregularly.

A marked feature of the deposits is the usually perfect cleavage and cross-cleavage, which break up the exceedingly hard and solid ore into blocks of irregular size, often resembling rough hexagonal prisms or pyramids, flat slabs, and other shapes. Some specimens of these are represented in Fig. 2, from a photograph.

It is owing to this kind provision of nature that this hard ore can often be mined without the use of explosives. The degree of perfection of these cleavage-planes, however, varies greatly; and, though never entirely absent, they are often so imperfectly developed as to be of no assistance in breaking the rock, in which case resort is often had to the time-honored

means of "fire-setting." In one instance the author has noted a set of these cleavage-planes running in regular concentric curves, as though the ore had been subjected to forces similar to those causing the lamination of schists.

Near the surface the ore is weathered; and this alteration and weathering commences along the cleavage-lines, so that these are often filled with minerals which may be taken as the result of the incipient decomposition of the ore. These are chiefly margarite and other talcose minerals, the formation of which along the cleavage-planes, of course, greatly assists the breaking-up of the mass. The extent to which weathering affects a deposit varies very much, owing to the varying composition of the ores, and probably also to the extent to which they have been previously cleaved. In some cases, weathering has proceeded so far in depth as to make the ore useless for the market; in others, some of the best ore is got quite close to the surface.

The facts here cited may be of some help in speculating on the origin and formation of these deposits, which are of interest in proportion to the mystery surrounding them at present. The author's suggestion is, that during the metamorphosis of the original, probably impure clayey limestone, these clayey and irony impurities segregated out, and were transformed to the present emery deposits. The exceeding purity of the present crystalline limestone, the apparent isolation of the deposits from any subterranean circulation, and the fact that the decomposition of the emery produces just that irony clay from which this theory traces its origin, all point to this as a feasible explanation of their formation.

(2) *Deposits of Emery Detritus.*—These have so far formed the chief supply of the emery-market, since, being on the surface and easily worked, though more limited in extent, they have been exploited in preference to the rock-emery. They consist of more or less rounded fragments of emery of all sizes, up to large boulders, imbedded in a compact clay, colored in some cases a bright red by the oxidation of the iron in the ore, and filling surface-depressions and pockets in the limestone, usually shallow, but sometimes over 20 feet deep.

Sometimes they occur by the side of deposits of rock-

emery, from the weathering of which they have evidently resulted; at others, no trace of the original deposit is visible, it having been either entirely weathered away or covered up by the detritus.

Quality and Value.—The value of the emery brought to market varies at present, according to the quality, from £2 10*s.* to £3 15*s.* per ton f. o. b. at Smyrna or other ports. The way the quality is judged at the mines is more or less empirical. The color and sheen of a fresh fracture, the character and degree of crystallization, the roughness of feel of a fresh fracture, distinguish the marketable emery from that discarded, and serve to classify the different qualities sold. The nearest approach to a systematic test is that originally devised by Dr. J. L. Smith to determine what he called the "effective hardness" of the stone. This operation is now carried out in the following modified form: The emery to be tested is powdered and sieved in a nest of sieves. What passes through a No. 50 mesh but remains on a No. 90 is taken. Of this, half a gramme is weighed out. Then a piece of glass, about 2½ by 3 inches in size, is accurately weighed and placed on a large piece of paper. The weighed emery is placed on the glass and gently rubbed with the bottom of an agate mortar for a fixed time (17 minutes), the powder that falls off on to the paper being returned to the glass at intervals. At the end of the 17 minutes the glass is brushed clean and weighed, the loss in weight in milligrammes representing the effective hardness of the emery. Thus the best Naxos stone is said to cause a loss of 80 milligrammes, whereas the worst qualities go as low as 25.

It is apparent, however, on the face of it, that this method must be very unreliable, and can only give an approximating clue in very experienced hands. A little more or less pressure exerted in rubbing, a little more or less time lost in restoring the powder to the glass, alters the result. Better was the original method of Dr. Smith, who rubbed a small sample of emery to an impalpable powder, without any regard to time.

In order to determine, if possible, a ready means of testing the quality of emery on the spot, the author made some tests with typical stones which gave the following results:

	Localities of Ore.					
	Gumush Dagh.	Serakuê.	Alaman.	Ak Sivri.		
				No. 2.	No. 5.	No. 6.
Specific gravity	3.92	3 91	3.899	3.82	3.87	3.87
Soluble in aqua regia, per cent	16	17.5	18.5	24.5	23	27

The ores are arranged in the order of their commercial values, and it appears that both the specific gravity and the solubility coincide fairly well with the abrasive quality of the ore. The difference of specific gravity is, however, too small to be a useful rough test. The solubility by itself is not reliable, but should give a useful indication, taken in conjunction with other physical properties of the stone. The "effective-hardness" tests gave no reliable results, constantly varying in the author's inexperienced hands. The only satisfactory way of testing the quality of the emery thus still remains the old method of measuring its abrasive power in bulk at the emery-mill, as is done now; and the mines will have to continue to rely on the verdict of value put on their stone by the buyers. That this is often governed by fashion is proved, to quote one instance, by the fact that because a certain good quality purchased once was of a red color, this accidental accessory, a red color, was always insisted on, or at least preferred, though other (blue) emeries were superior in hardness.

Working.—The emery is quarried or mined in tunnels (see Fig. 3) opening into big caves, the roofs of which are supported by pillars left standing. It is then picked over by hand (see Fig. 4), and the good emery is taken on camels, carrying about 4 to 5 cwt. each, to the nearest port or railway-station, sometimes over 20 miles distant. The transport is sometimes a greater expense than the mining, duty, etc. The royalty to the Government is about 13*s.* per ton, which amounts on the best ore to 17 per cent., on the low-grade ore to 26 per cent. on the f. o. b. value—a heavy tax.

2. *Chrome-Ore.*

This ore was also first discovered in 1848 by Dr. J. Lawrence Smith, near Broussa, in the province north of Aidin.

It occurs here, as elsewhere, exclusively in the serpentine, in the shape of pockets and veins of irregular extent and size. Its chief occurrences are marked on the map, Fig. 1, but in the district under notice its exploitation is practically a monopoly in the hands of Messrs. Patterson, whose mines are in the neighborhood of Makri.

The value of the ore depends on its contents of sesquioxide of chromium. Shipments have been made from Makri containing as much as 58 per cent. The lowest content marketable is 47 per cent., and this only if the ore is soft and easily crushed. Such ore has always been preferred to the very hard ore, which latter must contain at least 50 per cent. to be marketable. At present 50-per-cent. ore fetches £3 12*s*. per ton f. o. b. on the coast; 52-per-cent., £3 14*s*., and so on. In the early days of the industry good ore was worth about £12 per ton.

It is a curious fact, of which Mr. Patterson has assured me, that the best ore is got near the surface, and that in depth it invariably becomes poorer.

The concentration of low-grade chrome-ore will be a problem that will have to be faced in the future, when the rich deposits become exhausted, and attention might with advantage be directed to it now.

When the industry was at its height over 30,000 tons of chrome-ore were exported per annum from Makri alone.

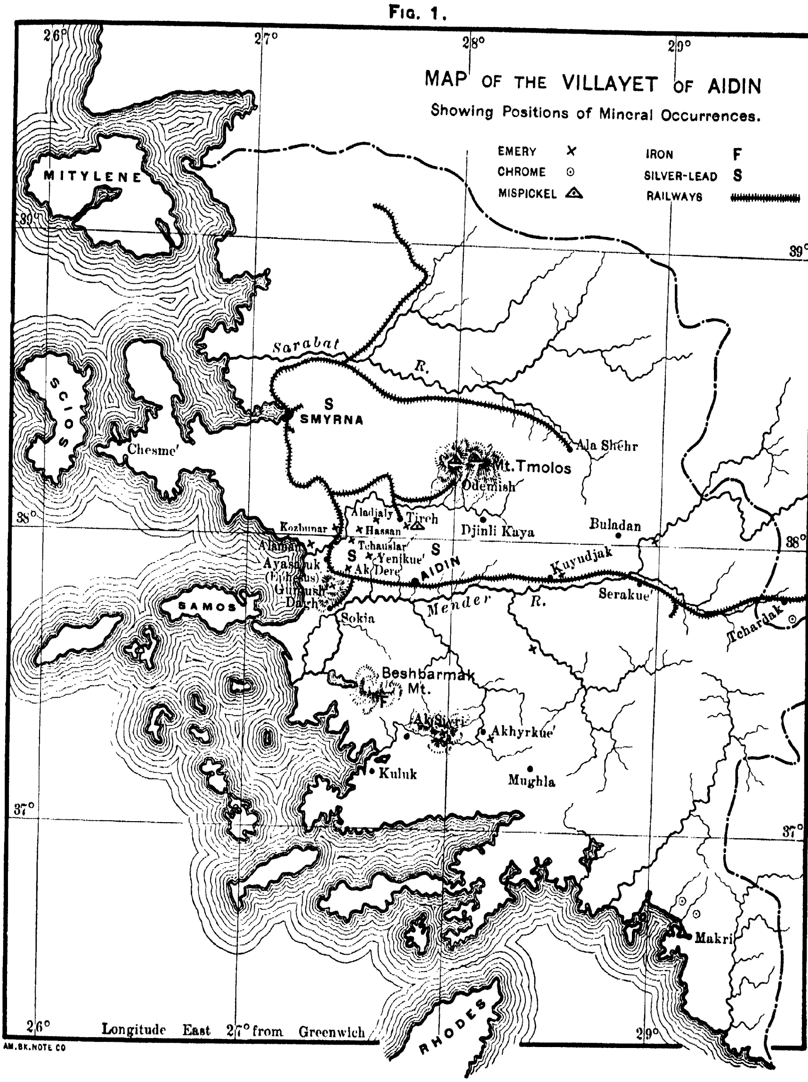
The ore is mined and handled much in the same way as emery.

3. *Auriferous Mispickel.*

In certain districts, as marked on the map, chiefly in the neighborhood of Mount Tmolos, numerous occurrences of mispickel, usually more or less auriferous, are found. The deposits examined by the author were all very irregular in shape and extent, following the bends and contortions of the micaceous schists in which they run, sometimes gradually merging into them and disappearing or swelling out suddenly.

The largest vein observed was about 15 feet wide, of pure quartz, with a few small stringers of mispickel, but most were very much smaller, though rich in mispickel. It would appear that during the process of crushing and straining which the schists underwent, irregular cavities, or at least lines of weak-

FIG. 1.



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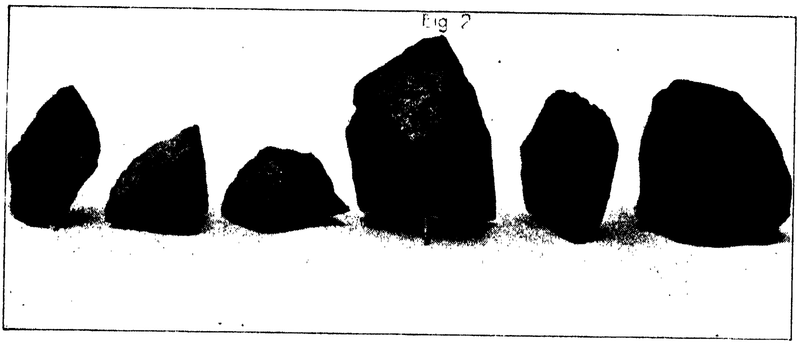
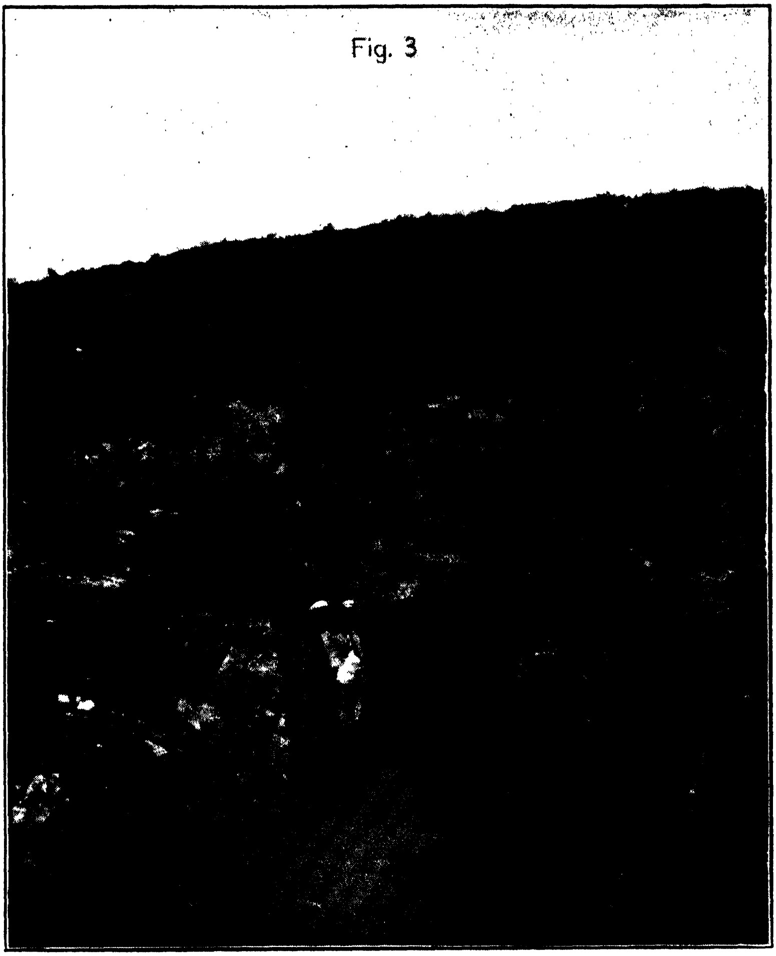
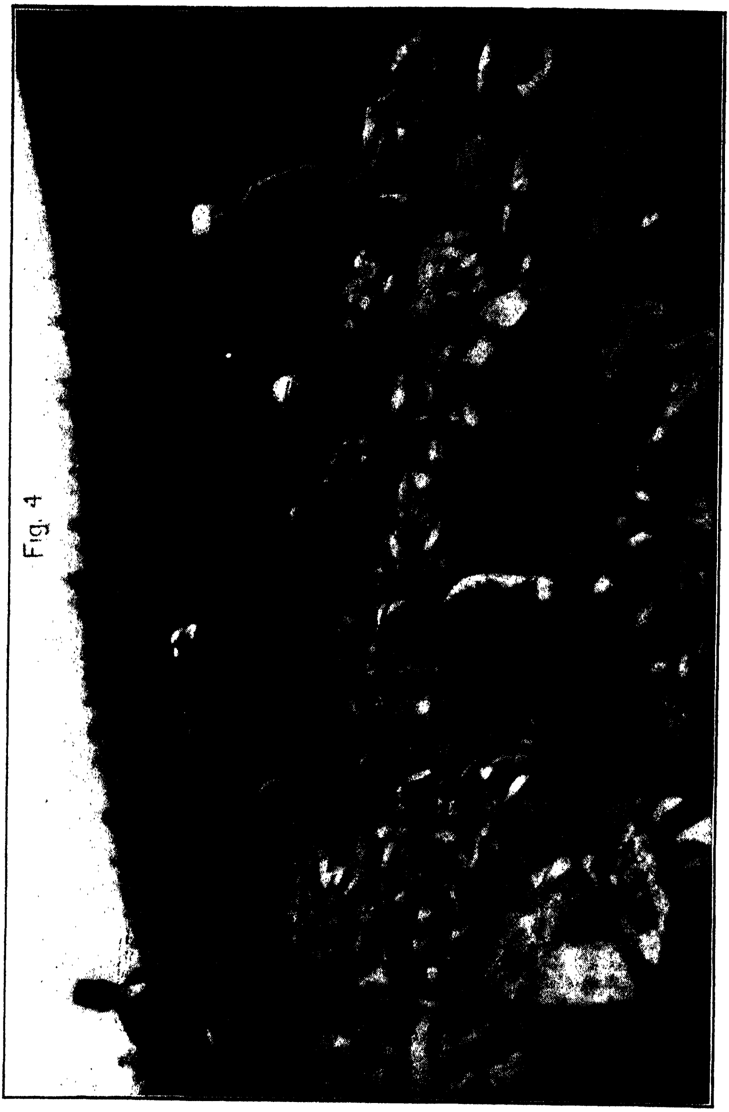


Blocks of Emery, as it Breaks in Mining.



Tunnel on Rock-Emery.



Chipping and Sorting Emery.

ness, open to the circulation of mineralized waters, were formed along the planes of lamination, which were widened and filled with quartz and mispickel deposited from solution. Everywhere in the schists these little lenses, veins, and sometimes regular reefs of pure barren quartz can be seen; but only in certain districts is mispickel associated with it.

The gold-contents of the deposits varied from a trace to over 3 ounces. None of them were of sufficient size or permanency for profitable working, but the country is apparently so impregnated with mispickel that there appears to be no reason why larger deposits should not exist; and if true fissure-veins cross the country, the chances are that they would also be filled with this auriferous mispickel in association with quartz, perhaps sufficiently rich to be worked.

4. *Antimony.*

At Djinli Kaya, about 10 to 12 miles S.E. from these mispickel occurrences, two antimony-lodes cross the country, one of which has been worked for some time. It is fairly regular in strike and dip, but varies greatly in width, being sometimes a mere stringer. It carries antimonite and iron pyrites, disseminated in quartz and soft crushed shale. In one place it has bunched out, forming a pocket of almost pure antimonite, from which about 4000 tons were extracted; otherwise it has not been of any considerable size. All the good ore has been taken out, and what is left now could only be made available by concentration, to which the large amount of associated iron pyrites is a serious obstacle. Of course, the possibility that further development might discover other rich bunches of ore is not excluded.

5. *Argentiferous Lead.*

The occurrences of this mineral are numerous throughout the limestone, but, unfortunately, most insignificant in quantity. At Yenikué, between Tireh and Balladjik, there are small quartz-veins traversing the limestone, carrying here and there little blebs and patches of galena, which, when carefully picked out, assay up to over 300 ounces of silver per ton; but the veins are insignificant, and their galena-contents still more so. Other occurrences in the country north of Aidin, and in

other parts, all small and patchy, are also poor in silver, assaying only up to 6 ounces per ton. At Kavaklideré, near Smyrna, there is a deposit of baryta carrying small grains and patches of galena, likewise too poor to work profitably. These galena-deposits occur generally where the crystalline limestone alternates with bands of micaceous and hydro-mica schists, and, though chiefly in the limestone, are sometimes found at the contact. Such country, generally surveyed, looks promising for mineral occurrences, and these poor prospects are disappointing. In the author's opinion, the entire absence of volcanic rocks, either in the form of dykes or sheets—a fact which constitutes the main difference between these and rich silver-lead districts in other countries—is accountable to a great extent for this want of concentration and richness in the deposits. Moreover, faults are very scarce, and true fissure-veins were not encountered.

6. *Copper.*

The same remarks apply to some so-called copper-deposits visited in the neighborhood of Ak Sivri. A little further south another copper-occurrence was reported, but not visited.

7. *Iron-Ore.*

There are some deposits of iron-ore on the Beshbarmak mountain, south of Smyrna, showing a mixture of hematite and magnetite running in micaceous schists, which might on further development prove worth working. Manecsa, an important town 28 miles N.E. of Smyrna, is the ancient Magnesia, famous for the loadstone (magnetite) once obtained from the surrounding mountains. The name magnet is supposed to refer to this locality.

8. *Gold-Ores.*

The occurrence of these seems to be limited. The northern slopes of Mount Tmolos, close to the ancient town of Sardis, where history or tradition places some rich ancient gold-mines, it did not fall to my lot to visit. On the north of the Bay of Smyrna, however, extensive ancient workings have been discovered, which are supposed by some to have yielded part of the wealth of Crœsus. To give a full description of them here would go beyond the limits of this paper. Suffice it to say that they have not yet been fathomed, though a vertical depth

of 200 feet below the crown of the hill has been reached. The country-rock is a trachyte, very hard in depth. What ore is left in the old workings now is completely oxidized, and consists of quartz with much iron oxide. Observations of other small veins in the trachyte leads to the conclusion that the ore must have been similar to that found in these, which is very complex, consisting of quartz, carrying argentiferous galena, blende, iron and copper pyrites, gold, and sometimes antimonite. The quantities of these minerals in the small veins were small, an average sample of from 1 to 2 tons giving: lead 7.6, copper 2.2, and zinc 2.7 per cent., and gold 13 dwt. and silver 5 oz. 13 dwt. per ton. It is to be presumed that the ore worked by the ancients was considerably richer, but it is a matter of surprise that they should have been able to treat such a complex ore profitably.

With regard to ancient workings in this country, and their bearing on any possible industry to-day, it should be remembered that the metals were of much more value in their day than in ours, and a very limited output was probably looked upon in those ages as untold riches. Also that, with their slave-labor, mining cost the ancients very little, and that their knowledge of metallurgy was probably far greater than we usually give them credit for. It may therefore be taken for a fact, established so far as experience has tested it, that in their old mine-workings there is nothing now left worth going for. Whether mining beyond the region of their work would now be profitable can only be told by testing each case individually on its own merits. In another district, a solitary instance, the Balia silver-lead mine, has proved fairly successful. Such an enterprise involves, however, the outlay of considerable capital in deep sinking—a risky investment, which, under present conditions in the Turkish Empire, it is difficult to recommend.

9. *Other Minerals.*

Other minerals occurring in the district under notice are: pyrolusite, cinnabar, realgar, and fullers' earth; but none of these have so far been found in sufficient quantity to give scope for profitable working.

General Conditions of the Country Affecting the Mining Industry.

Money.—The currency in vogue in Turkey is perhaps the most complex in the world. The standard coin is the medjid or dollar, worth 3*s.* 4*d.*, of which 5.4 go to the gold piece, the Turkish lira. The medjid is subdivided into piastres, and here worse complications commence. There are at least six different values for the piastre. There is the gold piastre, of which 19 go to the medjid; there is the piastre of exchange, of which 23.15 go to the medjid. In some cases the medjid equals 20 piastres; on the islands it is 25.05 piastres; at Balladjik and in the interior 22.5 piastres; in retail trade 33 piastres, called "tchuruk." Needless to say there is no coin to represent these values, the medjid being divided, instead, into 76 metalliks, of which there are 1-, 4-, 5-, 8-, 10-, etc., metallik pieces, besides the quarter and half medjids. Books are usually kept in piastres of exchange, and all accounts in other values have to be changed to these. Verily, book-keepers have no easy time.

Government.—The unsatisfactory character of Turkish administration is almost proverbial, and has scarcely been exaggerated. There is practically no security of tenure for the most legitimate mining enterprises. True, the Sultan's Firman for a concession once obtained is not to be disputed, but the difficulty is to get this. Without it, no matter how carefully all regulations have been complied with, claimants spring up to contest your rights and stop your work, and are actually encouraged to do so by the government. Does it not put untold backsheesh into the pockets of all officials connected with the law courts and the mining department? And how could these live without such means of getting paid? For years such claims are contested; from one court to another and back again the case is sent, and each time the previous verdict is reversed, unless one party refuses to bribe any more, when of course the other party is immediately successful—until some other claimant springs up. For anyone but a Turk to obtain a Firman is practically impossible, unless at the expense of far more money than the concession is worth.

Royalties to the government are in most cases very heavy, as has been instanced above for emery, which comes under the

heading of quarries. In mining proper, the nominal royalty is 5 per cent. of the gross value of the ore where it is sold, which may be in London or America; but really a deposit of £1 per ton must be paid before the ore can be shipped, the government kindly promising to return the balance over 5 per cent. on receipt of the bills of sale. Needless to say, if this is required in cash, the miner must wait indefinitely for it.

Duties are also heavy; but in this case the custom of bribing the officials has its advantages, as thus by judicious management sometimes only a small proportion of the real duty need be paid. In this connection it is amusing to note that the importation of dynamite is absolutely prohibited; yet a case is known where the government official of a district, for due consideration, not only closed an eye, but even actually sent an escort with a shipment of dynamite to a certain mine.

All these restrictions, however they may be circumvented, are nevertheless harassing and hampering to industrial enterprise, even when the prospect of a successful industry are very bright. It can easily be understood that where, as here, the mining industry requires fostering carefully to give it a chance of success, the oppressive policy pursued can only tend to destroy it altogether.

An Apparatus for the Removal of Sand from Waste-Water of Ore-Washers.

BY J. E. JOHNSON, JR., LONGDALE, VA.

(Atlantic City Meeting, February, 1898.)

THE description of the machine which constitutes the subject of this paper is best introduced by a statement of the conditions which led to its construction, which may be briefly given as follows:

The ore handled at the plant to be here described* is of the variety sometimes called by those familiar with the brown ores of the South, "Mountain," as distinguished from the "Lime-

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* Namely, that of the Longdale Iron Co. For description, see "Ore-Washer at Longdale, Va.," by Guy R. Johnson, *Trans.*, xxiv., 34.

stone" ore. The latter occurs chiefly in pockets and fissures in the surface of the limestone, generally as particles ranging from the size of a chestnut down, and scattered through clay of usually several times its own bulk. It is commonly found in the lower part of the mountain valleys in which it lies, has no regular formation, and contains very little sand.

The "Mountain" ore, on the other hand, has a very distinct formation, occurring as a regular stratum in the place of the No. 7, or Oriskany, Sandstone. It goes to depths of several hundred feet below the outcrop, and is frequently extremely solid, but in many places is in a gravelly state, and sometimes is mixed with a considerable proportion of clay.

This ore always contains a very much smaller proportion of foreign matter than the other. It yields about 5 tons of washed ore for every 8 tons of material sent to the washer.

Of the 3 tons of waste, about one ton is sand, which consists much more of very finely ground-up iron-ore than of ordinary silica, and is very heavy in consequence.

For many years it has been the practice of the Longdale Iron Co. to settle all the waste-water and tailings from these washers in large ponds, constructed for this purpose by dumping the slag from the furnaces (which is handled cold) to form high embankments running along the bottom of the valley from one foot-hill of the mountains to another, and lining the slope of the embankment with clay.

Ponds several acres in extent, and 20 to 30 or more feet in depth, are thus formed, and the water from the washers filters through the banks and comes out perfectly clear.

It is highly desirable to remove the sand from the waste-water, because the cost of these settling-ponds is considerable, and running both the sand and the clay into them not only fills them up one-half faster than the clay alone would do, but, on account of the weight of the sand, it is necessary to give the waste-trough a much greater inclination to carry it off than would otherwise be necessary, and when the length of the waste-trough exceeds 1000 feet, as at present, the height so lost reduces the available capacity of the ponds very materially. For several years this last fact has had the effect of necessitating the presence of one or two men to shovel sand from the trough, and one to remove it with a wheelbarrow or scraper, this being

regarded, however, as only a temporary expedient, and largely justified by the prospect of treating the sand thus moved (and stored), so as to recover from it a large proportion of workable ore.

These 3 men were not able to handle over 17 or 18 tons per day in this way, and it was not considered advisable to arrange the shoveling-place immediately over the track, and load the sand directly into the cars, on account of the slop, and the difficulties from freezing in the winter.

The removal of this sand by some more economical means seems a simple matter enough, but the obvious ways of doing it reveal, upon further consideration, serious objections, which do not appear on the surface. The simplest seems, at first sight, to be the use of two tanks, either with drop-bottoms or else arranged to tip when full, and return automatically when dumped, each being used alternately. But as these fill up, their area of cross-section diminishes, and the velocity of the water increases; and, since the energy of the water and its power to move sediment increases as the square of its velocity, it is evident that the percentage of sand deposited will diminish very rapidly as the tanks fill up, while to allow them to become only partially filled with sand before emptying would leave a large volume of water to dispose of without any effective means of handling it.

A chain-elevator, with its numberless joints, and the necessity of having the bottom-sprocket work under water, would not be satisfactory for so abrasive a material as sand.

A vertical wheel with buckets would be difficult to design so that the sand should be carried laterally far enough to clear the side of the trough without dipping up a great deal of water and spilling both sand and water on the shaft and running-gear as the buckets passed the top.

The machine finally designed for the purpose avoids all these difficulties, and is, besides, very simple in construction, and having given entire satisfaction since the first of July last, when it was put into operation, it is here described, in the hope that it may be of use to others.

Its general appearance is shown by Figs. 1 and 2.

The waste-trough, whose regular section is 13.5 by 12 inches inside, is enlarged into a tank or box of cast-iron, made in two

pieces and bolted together, 6 feet 2 inches long, 6 feet 10 inches wide at the top, and 19 inches deep, with sides sloping at 45°. This is extended at each end by trough-sections of sheet-steel, sloping both horizontally and vertically, to make a gradual inlet and discharge, and prevent eddies and excessive velocity of the water in the tank. This is a necessary condition for the deposit of entrained matter.

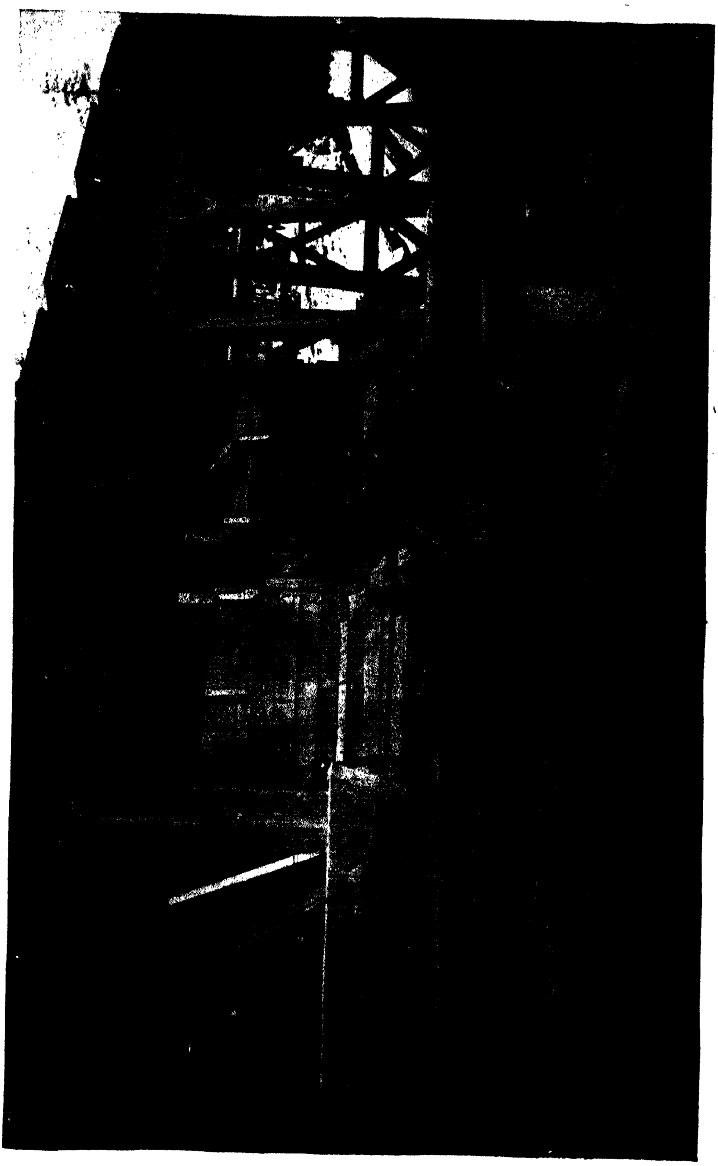
In this tank revolve two "shovel-heads," or wheels, whose axes of revolution stand at 45° from the vertical in opposite directions, meeting at the top, over the center of the trough.

The shafts of these terminate at the top in worm-wheels, both of which gear into one worm, revolving in a bath of oil in the center of the machine.

The shovel-heads each consist of a 6-armed "spider," whose arms stand at about 45° with its axis. Each pair of adjacent arms carry bolted to them, near top and bottom respectively, two 1.5-inch wrought-iron rods extending out a considerable distance past one of them. Both the rods of all three pairs extend in the same direction circumferentially, and the upper ones of all three pairs lie in the same plane perpendicular to the axis, as also do the lower ones. Each pair of these rods carries at its outer end a blade or shovel of sheet-steel, whose positions at oblique angles in all directions may be better seen from the drawing than described in words. It will be noticed that these blades or shovels are slightly bent along an approximately diagonal line, the rods which support them being bent also, so as to support them fully on each side of the bend. The blades are fastened to the rods with counter-sunk bolts. The effect of the angles at which the shovels stand, in combination with the inclination of the axis of the driving-shaft, may best be seen in Figs. 3, 4 and 5, which are respectively a section at the end of the trough, a side elevation with the side of the trough broken away, and a plan.

It will be seen that the blades, when at the bottom of their travel, pass close along the bottom of the trough in a position nearly horizontal, but with the leading edge inclined somewhat downward. When they have made a half-revolution they have been carried sideways far enough to clear the trough considerably, and at the same time have been raised some distance above it. They have also, during the same period, gradually turned from horizontal to vertical.

FIG. 1.



View taken from Left Side, Discharge-End, Diagonally.
(Sand-piles in cars showing at bottom.)

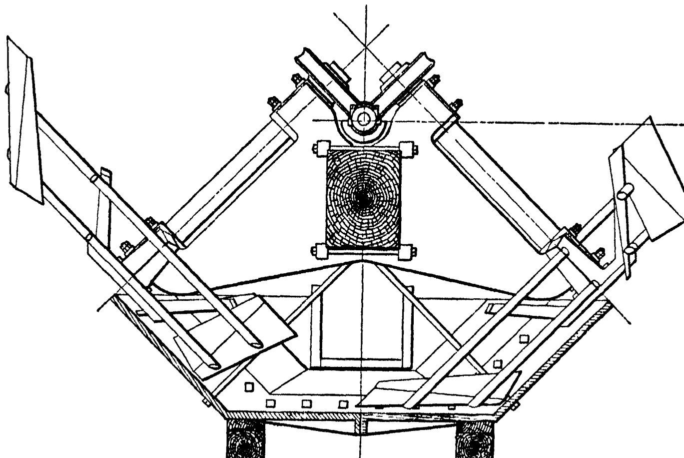
Fig. 2.

Machine stopped, as seen from Discharge-End.

The result of these motions is that they go into the sand exactly as a shovel does, pick it up, lift it clear of the water, pour off the water lifted on the shovel, carry the sand clear of the edge of the trough, and dump it, practically dry, into a car standing on the track beneath.

The machine, as shown here, differs somewhat from the original design. The principle has not been changed, but the shovel-head was originally very much larger, so that the blades were bolted directly to its sides; this being done on the supposition that the sand would pass over the blade without being raised by it, unless there were something at the back of the

FIG. 3.



Scale ⅜"=1 foot.

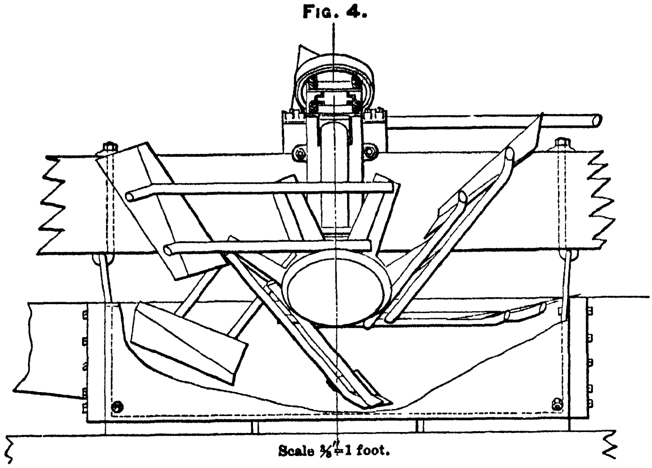
End-Elevation.

blade to stop it. The re-entrant angle between the blade and the side of the shovel-head thus really formed a bucket. This was soon seen to be a positive detriment, as the sand stayed out on the end of the shovel, while the water brought up when the bucket emerged ran down into the angle and poured out into the car, time not being given it to run out before the line of the bucket cleared the trough. Skeleton buckets were then made, and cured this trouble completely, but the sand showed a tendency to run off over the back edge of the blade before it quite cleared the trough, and the back edge of the blades was then bent up to delay discharge. The amount and line of bending here shown were adopted after one or two trials.

It was noticed that the sand seemed to "set," and became much more coherent, around two holes, which happened to be in the blades, than elsewhere; and it seemed possible that the use of perforated blades would have the same effect on the whole body of sand on the shovel by quickly draining away the entrained water, and so reducing the tendency of the sand to flow like a semi-fluid, this tendency to flow still causing the discharge to take place a very little too early.

This was tried with entire success, the entrained water drained away almost instantly, and the sand became so solid that it did not start to discharge until the blade was approaching the vertical position, while the water discharged with it was reduced to a minimum.

FIG. 4.



Side-Elevation, with Part of Box Broken Away.

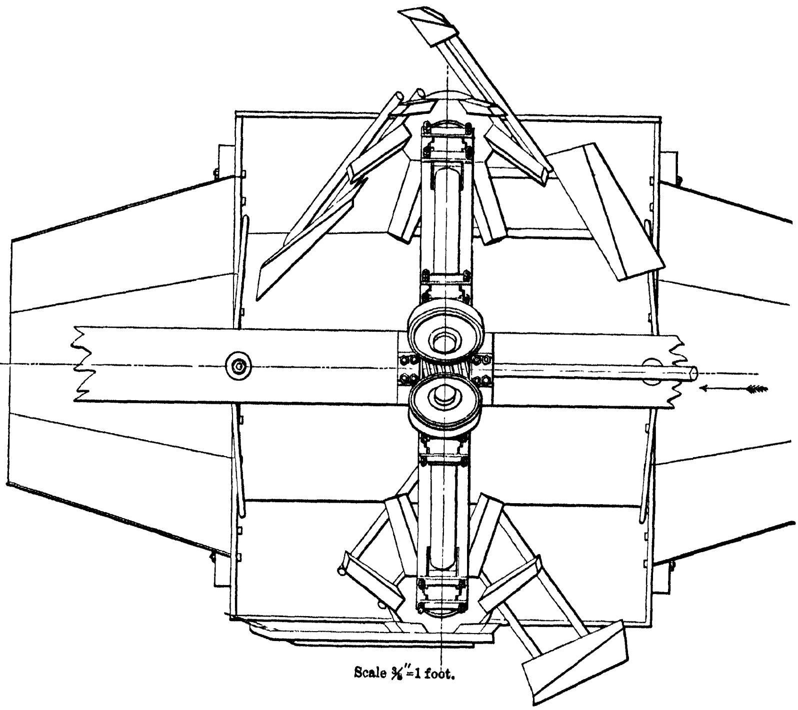
This is shown by the shape of the sand-pile in the car in Fig. 1, in which the tops of the cars just show at the bottom of the picture.

It being impossible to get sheets perforated with holes small enough to hold the sand, and at the same time thick enough to give the necessary stiffness, a heavy plate with coarse perforations was used as a backing for a thin plate with fine perforations; but subsequent observation leads to the belief that the perforations need not be so fine, since many of them fill up in time, and (owing to the firmness of the sand when the water drains away) very little sand passes through them even when they are considerably larger than the individual grains.

This being the case, plates with the size of holes required may be obtained thick enough to be as stiff as is necessary.

After the skeleton blades were found to be correct, the solid shovel-heads, which were very heavy and clumsy, and did actual harm by projecting down into the trough, thus reducing its effective cross-section, were thrown out, and the spider with

FIG. 5.



Top-View.

rods was adopted, which resulted in a considerable increase in the output.

The original speed adopted was six revolutions per minute, but this was found to be too high, agitating the water unduly, thus causing sand to be carried by, and sometimes slopping water out of the trough. This speed was reduced to three and a third revolutions, which entirely disposes of both evils.

The blades travel up-stream while in the trough, and the two shovel-heads are set so that the blades on each come half-way between those on the other. This makes the reduction of cross-

sectional area a minimum, and keeps it nearly constant, so that the flow is about as uniform as possible.

The machine is driven by a worm-gearing, as before stated, the worm-shaft carrying on its other end a 48-inch pulley belted with a 3.5-inch belt to the picking-table line-shaft.

The speed-reduction ratio of the worm-gear is fifteen to one, so that the shaft turns at 50 revolutions and the belt-speed is about 630 feet. This belt is never very tight, and the power consumed cannot exceed 2 H. P., and probably never reaches that.

It should be noticed that the machine is belted direct to the line-shaft (which is belted direct to the engine), without counter or loose pulley. The reason is that, if the washers could start before the shoveler, the box would be filled with sand and some of the shovels would be buried in it, so that the belt would be unable to start it. By belting direct, the machine starts when the engine does, and without difficulty.

The ordinary output of the machine is from 30 to 40 tons of sand per day, or just about twice that of the three men previously employed, the quantity of water passing it being about 700 gallons per minute.

The difference in coarseness of the material passing over into the settling pond and that removed is striking, and seems to indicate that the machine gets practically everything out of the water except the clay and fine silt.

The difference in the rate at which the settling pond fills up since the machine was put in is also very noticeable, while the least permissible gradient for the waste trough is reduced about one-half. No attention of any kind whatever is required beyond shifting out the loaded cars and putting in empty ones.

A few of the larger details are shown, but require very little comment. It may be noted, however, that the worm has a Schiele's footstep-bearing curve to take the thrust of the worm, which is quite heavy, on account of the small diameter of the worm-wheel relative to the radius of action of the shovels.

This has furnished, in service, a strong contrast with most worms, since it has never given a particle of trouble.

The main bearing-bracket has jaws slotted to take the ribs on the sides of the chilled boxes, the slots being much wider in

the direction of the axis than the ribs, to allow end-adjustment of the shovel-heads for clearance.

The main shafts are of cast-iron, flanged and bolted to the shovel-heads, as shown.

The worm is arranged, as shown, to run in oil, and not only lubricates itself and the worm-wheels, but enough oil trickles down the shaft to lubricate the main bearings also. But what contributes more than anything else to the ease of running of the machine is, that all bearings are completely removed from the possibility of getting sand or dirty water in them.

The method of supporting the machine is clearly shown, except the trestles at each side of the track, which carry the three sticks of timber shown. Of these the top one supports the bearing-brackets for the shovel-shafts, and. in fact, all except the settling-box, and has to be of large section to resist the torsional stresses put upon it by the fact that only one shovel at a time is working—first on one side and then on the other.

The practice of settling waste from ore-washers is so little followed that it is to be feared that this machine will have very little application in its original field. The chief justification for describing it is in the hope that it may find use in kindred industries elsewhere.

Note on Limonite Pseudomorphs from Dutch Guiana.

BY R. W. RAYMOND, NEW YORK CITY.

(Atlantic City Meeting, February, 1898.)

THROUGH the courtesy of Mr. James H. Mayo, a member of the Institute, who is in charge of the operations of the Mindrinetti Company in the Saramacca district of Dutch Guiana, I have received specimens of certain peculiar pseudomorphs of limonite after pyrite, which occur abundantly in the auriferous deposits of that region.*

I am indebted to Mr. H. Tweddle, a gentleman connected with the same company, for some particulars of interest con-

* These were exhibited in connection with the presentation of this note.

cerning the occurrence of these pseudomorphs, which may prove to be significant as to the nature of the formation in which they occur, and the origin of the gold contained in it.

All those which I have seen (and, I think, all which have been encountered) are cubes, excellently preserved, and, though rough on the surface, not water-worn so as to destroy the definite completeness of form. They are often aggregated by intergrowth of crystals; but never, so far as I know, are regularly twinned.

They vary in size; but the majority of them measure about 0.5 inch on the side. The largest one in my possession measures 0.75 inch. These dimensions are not exceptional for unaltered pyrite. Cubes up to 6 inches have been found in Elba; and very large crystals have occurred in some of the Cornish mines. But it is, I fancy, not usual that pseudomorphs, and especially pseudomorphs found in detrital deposits, exhibit such size; for the process of pseudomorphism would naturally favor the action of the mechanical agencies which, in such cases, tend to break down all but the hardest mineral structures.

These cubes are generally hollow, and show, when broken, an interior cavity, loosely filled (or partially filled) with an ashy powder. Sometimes, however, they are quite solid, and the interior presents a hard, red, amorphous ferruginous mass. In the one specimen of this kind which I have examined, there is still an indistinct line of demarcation between the interior mass and the outside crust. I am therefore inclined to believe that these solid pseudomorphs were once hollow, and have been filled by subsequent infiltration and precipitation.

These crystals and crystal-aggregates are found on or near the surface in an alluvial deposit, lying upon what seems to be a "bed-rock" of clay. The pebbles of the alluvium are generally quartz or iron oxide, while the matrix is clay, strongly impregnated with iron. The quartz is always sharply angular (not rounded by water-action); and the iron is usually limonite, but sometimes approaches red hematite. It is a curious circumstance that no unaltered pyrite has been encountered in the explorations hitherto made. The gold found in this alluvium is always "sharp," and does not seem to have suffered transportation for any considerable distance. Thus all the indications point to a very complete decomposition and oxidation

in situ of the pre-existing rock, and to the derivation of the gold from rich quartz-veins in the immediate neighborhood. Many quartz-veins are in fact to be seen; but they do not show free gold to any striking extent. At least, Mr. Tweddle informs me that he saw none in place in the vein-rock, though numerous specimens containing it, and said to come from the veins, were exhibited to him. It seems to me quite possible that the richer zones of the quartz-veins have been disintegrated to form the auriferous stratum, and that this disintegration has been checked at harder and lower zones, which constitute the present outcrop.

Perhaps, however, this "sharp" gold is the result of solution, transportation and final precipitation (*e.g.* by organic matter) on its present locality.

The prevalence of iron oxide throughout the auriferous layer certainly points to an original association of the gold with pyrites; and Mr. Tweddle has found several nuggets of gold, completely encased in iron oxide, which may, or may not, have been the product of the oxidation of the pyrite which once contained this particular gold.

Dr. A. R. Ledoux, New York City, has kindly made, at my request, an analysis of these pseudomorphs, and writes concerning them as follows:

I have made the following tests with the samples of altered pyrites which you handed me. You are well aware that pyrite readily alters to pseudomorphs, and that the mineral has been replaced by sulphate of iron, limonite, hematite, silica and graphite. Dana also mentions pseudomorphs of "ochreous clay," and numerous authors describe artificial pseudomorphs prepared in the laboratory. The samples you sent me are characterized by a distinct outer crust of a light red color, the interior being a darker red, honeycombed, or absolutely hollow, and containing a reddish powder. An analysis of the outer crust shows:

	Per cent.
Combined water,	5.90
Alumina,	17.70
Silica,	30.44
Sesquioxide of iron,	42.90

There are no appreciable amounts of lime or sulphuric acid free or combined. The crust is evidently a mechanical mixture of oxide of iron and clay. After removing the outer crust an analysis of the interior shows:

	Per cent.
Combined water,	7.50
Silica,	2.34
Alumina,	0.90
Sesquioxide of iron,	87.94

This interior portion is, therefore, practically a pure sesquioxide of iron.

Having been informed by you that the mineral was found in a gold-bearing gravel, I tested the samples for gold. There was none in the external crust, but the interior contained gold to the extent of about $1.00 per ton of 2000 pounds. A number of years ago I remember to have seen samples of hepatic pyrites from North Carolina, which, on being broken open, showed in the interior a distinct nucleus of gold, in one case as large as a pea, and I could not avoid the thought that the pyrites had crystallized around the gold as a nucleus rather than that the gold had segregated from the mass of the surrounding mineral. While no gold is visible to the glass in the samples you sent, it is interesting to note its presence.

It would be presumptuous to propound upon the limited evidence at hand a theory as to the origin of these deposits and the source of their gold, but the facts given above will sufficiently indicate that further study may develop a most interesting, and perhaps a highly complicated case.

A few general observations concerning the Dutch Guiana district, which is likely to become an important producer of gold, may be appropriately added, on Mr. Tweddle's authority. I make no apology for the incomplete and fragmentary character of these data. When the operations of the Mindrinetti Company shall have advanced to the stage of actual working results, it is to be hoped and expected that more detailed descriptions will be presented to the Institute, in accordance with promises received from the parties interested. Meanwhile, as Secretary of the Institute, and Editor of its *Transactions*, I take this opportunity to emphasize once more my sense of the value of preliminary notes, however partial and incomplete, and the unwisdom of maintaining entire silence until elaborate and thorough treatises can be prepared. To most of us busy engineers the latter occasion never comes, and we are in danger of remaining mute until we have also become unable to speak.

The main gold-bearing belt of Dutch Guiana crosses the country at about the 5th parallel of N. Latitude. Its width is unknown, the country being entirely unexplored. The natives say that there is another and parallel belt further south; and, though this statement has not been verified by any considerable practical exploration, it is inherently plausible, and, if true, establishes two auriferous belts which traverse French, Dutch and British Guiana, and enter Venezuela from the east.

In Dutch Guiana this belt presents low rolling hills, of slight altitude above sea-level. Most of the country is thickly wooded, with many water-courses and much swampy ground,

rendering the work of prospecting extremely difficult. There are occasional open spots, known as "savannahs," generally presenting fine, white, "sharp" quartz sand, and traversed by thick strata of "grit" or quartz-conglomerate. The rest of the region, from the coast inland for nearly 100 miles, is an alluvial formation, mostly of mud. At about 100 miles from the coast some stratified rock (slate) appears, but the bed-rock or country-rock of the mines is hard clay, generally ferruginous.

Mr. Tweddle thinks, from what he has seen of the region, that Dutch Guiana has suffered from denudation to a much greater extent than British Guiana. In the latter province there are numerous "table-mountains" of conglomerate (doubtless the result of still earlier erosion) which have been mostly, if not wholly, obliterated in the former. The water-shed separating the rivers of Dutch Guiana from the Amazon is consequently much lower than the corresponding divide in British Guiana.

In the Saramacca district, between the Saramacca and Surinam rivers, where the Mindrinetti Company is operating, the gold occurs in sharp quartz gravel, generally white (but containing ironstone pebbles), lying above or between strata of white and blue clay, which is in places a regular pipe-clay, and is doubtless the product of the decomposition of the slate of the original mountain chain, which has now almost completely disappeared.

Hitherto all the mining done has been in the richest creek-bottoms, and only ground which yielded more than $2.50 per cubic yard has been worked. Of course, the value of the deposit varies greatly, as does that of all gold-placers. At the place where the company referred to will begin operations, preliminary tests over a considerable area have yielded by ordinary sluicing about $2 per cubic yard *in situ*, while the tailings have shown, by fire-assay, values ranging all the way from 50 cents to $12 per ton. After all reasonable deductions for the inevitable inaccuracy of preliminary tests, these figures indicate a handsome margin of working profit. The country, being practically flat, affords neither "head" for natural hydraulic pressure, nor lower dumping-ground for tailings. The company has, therefore, adopted the "Lay" hydraulic mining system, and the result of its operations will thus be deeply interesting to mining engineers as well as to mining geologists and mining investors.

The Relation of the Strength of Wood under Compression to the Transverse Strength.

BY B. E. FERNOW, WASHINGTON, D. C.

(Atlantic City Meeting, February, 1898.)

About eight years ago a comprehensive study of American timbers was begun in the U. S. Division of Forestry with a twofold object. On the one hand, it was desired to determine the working-qualities of our numerous economically important timbers, and to furnish reliable standards of strength. On the other hand, it was intended to get a clearer insight into the properties and behavior of wood in general, with a view of establishing more rational methods in its use, and to furnish data for physical inspection which would permit an intelligent application of the standards of strength, and place upon a rational foundation the so-called "factor of safety," which is at present a mere baseless guess.

The first object, namely, the establishment of standards for any given species, requires a very large series of tests, since wood is an exceedingly variable substance, so much so that in the same tree a variation of 50 per cent. in strength may be found; hence the greater need of rational data of inspection.

So far, the insufficient appropriations and facilities at our disposal have permitted us to establish really reliable standards for five species of southern conifers only. These have been deduced from nearly 20,000 tests. Besides these, 10,000 other tests have furnished indications regarding some two dozen other species.

This work is now entirely abandoned, waiting for more prosperous times.

The study of wood in general, however, can be carried on with smaller means, and has this year yielded not only very interesting, but highly important results, which will soon be published in detail.

It is the purpose of this paper to make to the Institute the

first announcement of a few of these results, and especially of one—a discovery which will place the engineer in a better position than he has ever yet occupied in the designing of wooden structures.

It is well known that the most satisfactory and reliable test of wood that can be made is that of endwise-compression. The crushing-load is directly related to the cross-area of the compression-piece, and therefore the unit of strength is at once expressed by division of the former by the latter. On the other hand, the transverse test of a beam (the form in which wood is most frequently used) is, in the first place, a compound test, combining compression, tension and shearing; and, in the second place, in order to get from the test a unit-expression, a formula is introduced, the correctness of which has been doubtful, and which is now known to be incorrect, but which, nevertheless, is still used by engineers.

Now, the important discovery which has resulted from the study of our large series of tests is that there is a direct relation between the strength in endwise-compression and in cross-bending; in other words, that the transverse strength of a beam can be directly computed from its compression-strength.

The credit for this discovery, and for the development of the formula expressing it, belongs to a young engineer, Mr. S. T. Neely, who was charged with the duty of compiling our test-data for use.

Without entering into any mathematical details, a few statements will show the course of reasoning and its results.

1. It is established by theory and by extensive experiments that, in a beam loaded in the middle, the distortions are not only proportional to the load, but are equal on the compression-side and the tension-side of the neutral plane until the elastic limit has been reached.

2. From the study of strain-diagrams and data of several thousand tests it appears that the elastic limit of a beam has been reached when the extreme fiber-stress becomes equal to the compression-strength of the material; in other words:

3. The cross-bending strength at the elastic limit is practically equal to the compression-strength at failure, and we need only determine the latter to have at once a safe basis for the designing of wooden beams.

This is the practical result: That the necessity for tests is reduced to a minimum, and no doubtful formula need be introduced in utilizing the test-data. Since no beam should be designed to be strained beyond its elastic limit, nothing but the compression-strength needs to be known for practical purposes.

Nevertheless, Mr. Neely has followed up the further behavior of the beam to rupture, has developed methods of determining the position of the neutral plane at any time until rupture, and has also shown how to calculate from a compression-test the beam-strength at rupture.

Other important results and deductions from these series of tests may be stated as follows:

1. Wood-testing should be done preferably on green or soaked timber, thus eliminating the variable influence of moisture on strength, which begins to assert itself when the moisture is below 32 per cent.

2. A well-planned series on small laboratory-sizes furnishes more reliable standard and average values for practical use than tests on large beams and columns. This conclusion is the result of some 60 tests on large beams and over 100 tests on large columns, compared with over 1000 tests on small pieces cut from them.

3. The best size for compression-pieces to furnish the most uniform results is a cube of 2 or 3 inches.

4. A difference of 10 per cent. in compression-strength values for conifers, and of 20 per cent. in hardwoods, cannot be considered as a valid difference for practical purposes, being chargeable to the natural variability of the material.

5. Wood compressed across the grain increases in strength comparatively, and does not lose its strength in endwise-compression, pieces compressed to 50 per cent. of their original height exhibiting as much compression endwise-strength as uninjured ones.

Sectional Cushioned Rolls.

BY JOSEPH WILLIAM PINDER, ELKO, NEVADA.

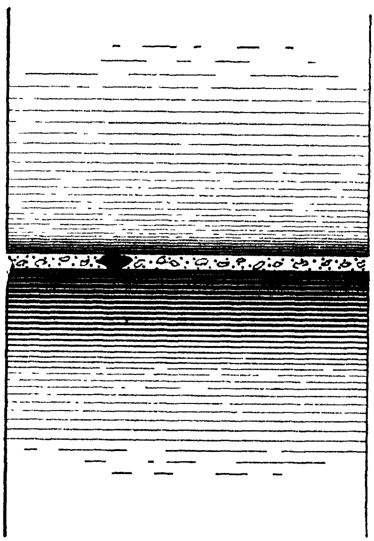
(Atlantic City Meeting, February, 1898.)

EVERY millman engaged in the operations incident to the handling of crushing-rolls knows that in ordinary practice, when fine product is desired, the ore-materials delivered to the machine, divided into four sizes, may be estimated, approximately, as follows:

	Per cent.
Passing a 4-mesh screen,	40
" 2- " "	30
" $\frac{3}{4}$-inch-mesh screen,	20
Larger,	10

This estimate refers more particularly to the requirements of medium-sized rolls and fine crushing.

FIG. 1.



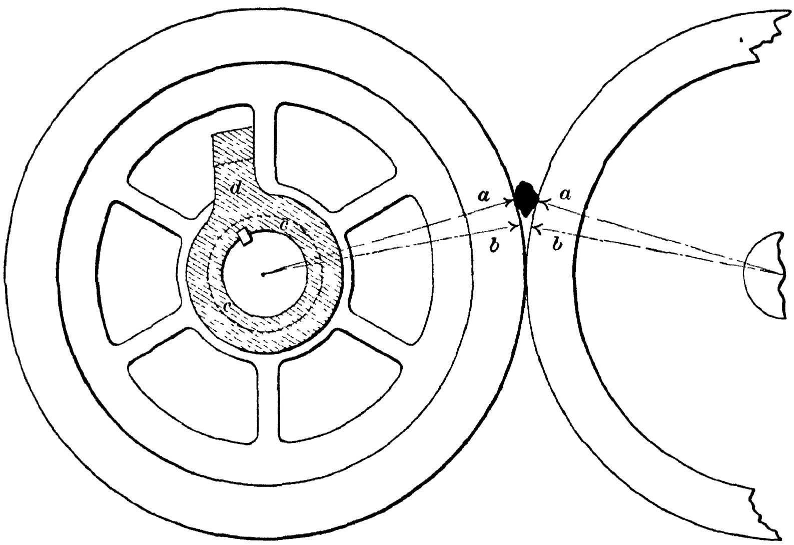
Large Piece Passing Between Ordinary Rolls.

But, depending largely on the character of work to be performed, the coarser material is often larger and of a greater percentage, while the bulk is always more or less fine.

As the common Cornish or other crushing-rolls are now operated in ordinary plants, it is almost impossible, economically, to regulate the first operation of preparing the ores for the rolls so that any approach to a uniform size may be obtained. Owing to this unevenness in size it is always necessary to pass and repass the same material several times through the machine in order to make a successful operation.

As the ore is fed into the rolls, when the larger pieces are clutched, and the pressure begins to bear upon them, the first strain on the springs tends to separate the rolls (Fig. 1 and

FIG. 2.



Large Piece Passing Between a Solid and a Sectional Cushioned Roll.

Fig. 2, *a*) until the crushing-point, or point of greatest resistance, is reached (Fig. 2, *b*) before the pieces are actually crushed. In this way, especially in crushing hard ores, the rolls are forced apart as much as 60 to 80 per cent. of the time. This parting is not infrequently as much as half an inch and more.

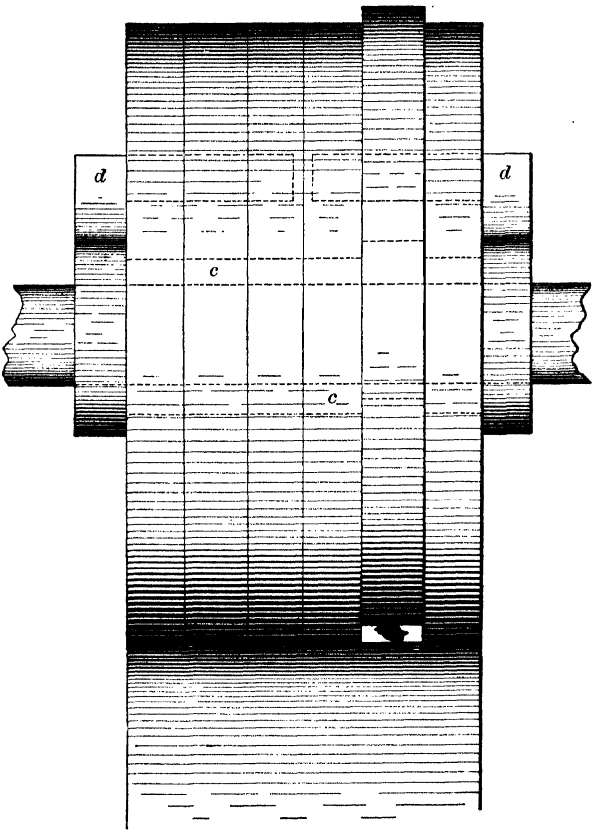
By far the greater part of the material being, as I have said, fine when it reaches the machine, has a tendency to pass through the rolls when thus opened by the larger pieces, and so to escape the desired grinding (Fig. 1). The same may be said of coarser work, the relative proportion of sizes being the same.

In order to correct this defect as much as may be, and in-

crease the efficiency in crushing the finer material that would otherwise pass the rolls, as I have shown, I have devised and practically tested a sectional cushioned system of rolls, applicable as well to all mills using rolls or rollers as a means of crushing and grinding ores of all kinds.

The system, as applied to the Cornish, or ordinary rolls of

FIG. 3.



Elevation of Sectional Cushioned Roll.

that class, consists in the division of one of the two rolls into several sections (four, six or eight), and the introduction of a stiff rubber cushion for each section, fitting snugly into the bore of the hub, and closely around the shaft which passes through it. This cushion is made of the stiffest car-spring rubber, from $\frac{3}{4}$ inch to 1 inch thick, according to the size of the machine (Figs. 2 and 3, *c*).

This sectional roll receives impulse from two steel arms, one on each side, keyed to the shaft, each arm passing through and pressing against the spokes of one-half of the sections (Figs. 2 and 3, *d*).

In every other respect the machine remains the same as usual, except that the main springs should be stiffer.

In the operation of this system it will be clear enough that when the larger and harder pieces of ore fall into the rolls, and the strain on the springs in the act of crushing causes the rolls to part, only that section in contact with such a piece will be parted until the necessary crushing-pressure is reached, after which it will spring back into place, while all the other sections will continue to crush such finer material as may fall in contact with each, which might otherwise pass between solid rolls without crushing.

Experiment has already proved more than 30 per cent. increased results.

When larger rolls are used, and coarser product is desired, the cushioned rolls, of course, may be set to crush to any size. With broader face to the rolls, and an even, well-distributed feed, the increase in capacity is very great.

The application of the sectional cushioned system to such pulverizing-mills as the Huntington, the Bryan, and others, gives great increase in output.

A United States patent has been applied for.

A New Form of Ingot-Mould for Casting Brass or Bronze Ingots, with Remarks on the General Form of Ingots.

BY ERWIN S. SPERRY, BRIDGEPORT, CONN.

(Atlantic City Meeting, February, 1898.)

Brass or bronze chips, grindings, buffings, washings, and miscellaneous scrap metal sooner or later find their way into the hands of the so-called metal-smelter or "refiner," whose refining or smelting operation consists simply in melting the material, adding the requisite amounts of other metals, and then casting it in the form of *ingots*, which are afterwards sold

for making brass castings. Phosphor-bronze, aluminum-bronze, manganese-bronze, and many other alloys already mixed, are sold to the consumer in ingot form, suitable for remelting without the addition of any other ingredients.

Such ingots vary somewhat in size and weight, depending upon the notion of the maker, but are contained within the limits of from 15 to 25 pounds each, the idea being to obtain, for introduction into the crucible, as large a mass of metal as would not be too unwieldy to handle.

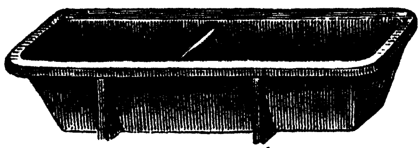
With brass "run down" from scrap the appearance of the ingot is not material; but with special alloys the metal-mixer strives to obtain as clean and perfect ingots as possible, the value of the product often being enhanced thereby. In order to produce such ingots it is necessary to cast them in a metal mould, one of iron being universally employed. In the manufacture of copper-ingots it is customary to use copper moulds for casting, on grounds of economy, and for the production of a better surface on the metal; but in carrying out these ideas in the brass industry it is obvious that it would be necessary to make the moulds of the same alloy as that they were to receive. Unless, therefore, the temperature of the molten metal were nearly at the solidifying-point (as it is with copper when poured) the moulds would be attacked. (Indeed, even iron moulds are often attacked by certain alloys, particularly phosphor-metals, the metal occasionally penetrating to a considerable depth, if allowed to strike in one spot.) Applying "dope" or oil to the surface of the mould lessens, but does not remove, the difficulty, and it is often impracticable to cool the metal before pouring.

For such reasons a mould either of cast or malleable iron is employed. The form generally sold by dealers in brass-founders' supplies is shown in Fig. 1. The ingot produced by this mould is approximately 3 by 3 by 7 inches in size.

The old adage, "Let every tub stand on its own bottom," could be well applied to such moulds, as anyone who has had occasion to use them can testify. With small crucibles the trouble is seldom experienced; but when pouring from a large crucible the mould will tip up and the contents will be spilled. Casting by resting the crucible on the side of the mould is out of the question, if more than a few ingots are to be cast. The

difficulty can be partially overcome by carrying the projections *a* and *b* (Fig. 1) around to the end; but even then other difficulties are encountered, which are very troublesome. When the ingots have cooled the moulds are inverted and the bottom is struck with a hammer, causing the ingot to fall out. As in the average case one cannot wait for the whole mass to cool, so that it can be handled, it becomes necessary to grasp the mould with tongs and invert it, an operation easier to

FIG. 1.

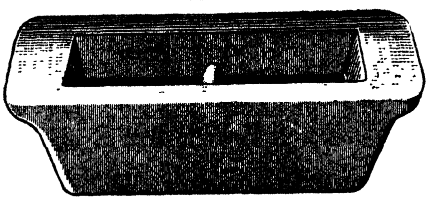


describe than to carry out, especially if the mould has been filled to the edge, for then there is nothing to grasp.

In order to overcome this difficulty some makers produce a mould with a lip on each end for grasping with the tongs, as shown in Fig. 2.

This lip enables one to grasp the mould with the tongs and quickly invert it. As ordinarily constructed, however, the lips are left smooth, so that it is difficult to obtain a firm hold

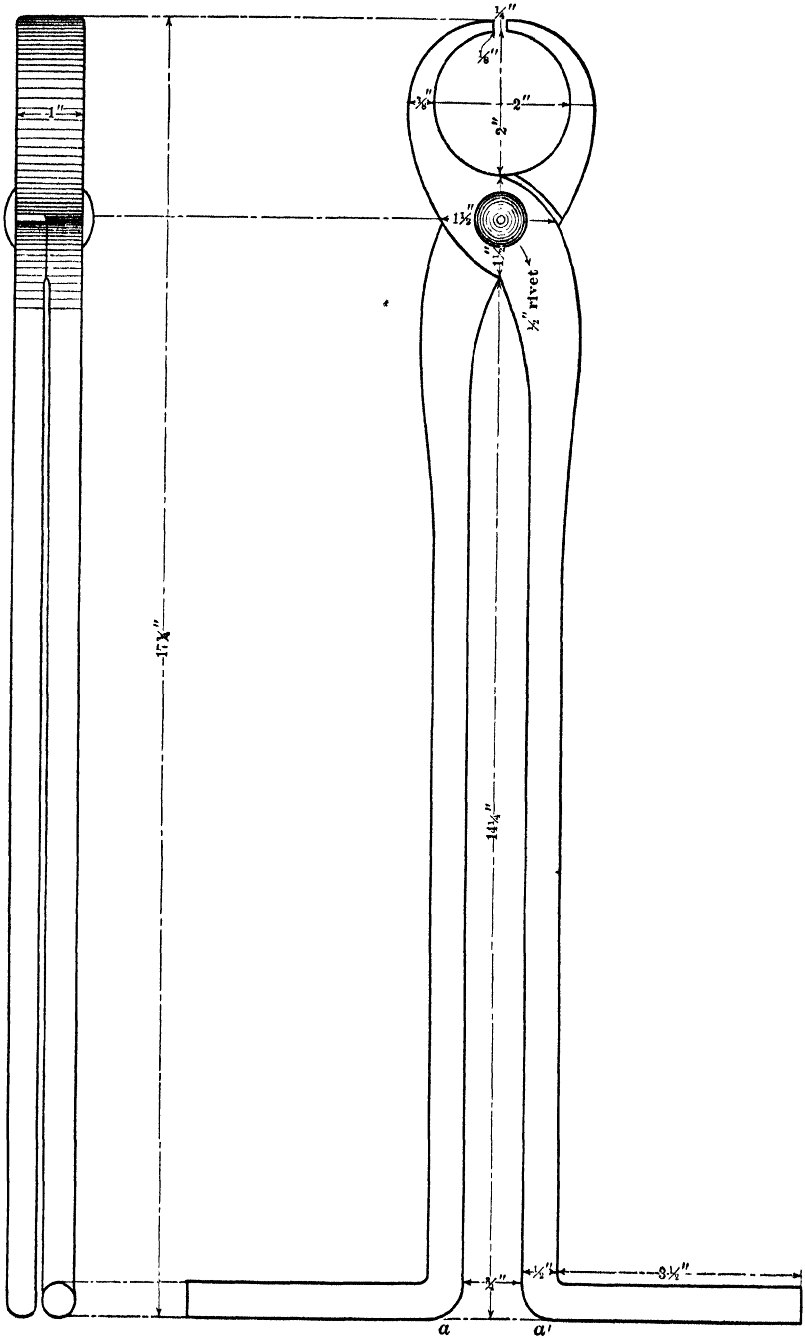
FIG. 2.



while dumping the mould; it is well, therefore, to leave a pocket in the under side of the lips to prevent the tongs from slipping.

The tongs employed for handling the ingot-moulds are shown in Fig. 3. If desired, the handles can be cut off at *a* and *a'*, leaving them without the bend, but after a trial of both kinds the author prefers that given in the drawing. They are useful both for dumping the moulds and for handling the ingots while hot.

FIG. 3.

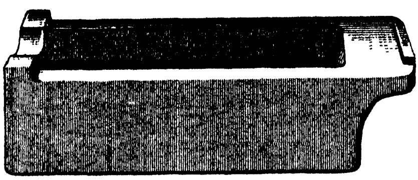


INGOT-TONGS
Weight, 3½ lbs.

The extended lips on the mould shown in Fig. 2 necessitate a rest being placed in front of it as a support for the crucible, for in this pattern the danger of upsetting is increased by the extension. As considerable trouble was experienced in the use of such a mould, the form shown in Figs. 4 and 5 suggested itself. This design has been used by the author for several years with excellent results. There is no possible danger of the mould tipping, and a rest in front is unnecessary. In addition to the pocket on the under side of the lip, a ridge was placed on top as an additional grip for the tongs, thus preventing them from slipping.

The author has found soft gray iron to answer the requirements better than that of a hard, close-grained nature. Moulds of the latter material often crack when used only a few times. Malleable iron is often preferred, and the mould shown in Fig. 1 is designed for such material; but if the proper sort of gray

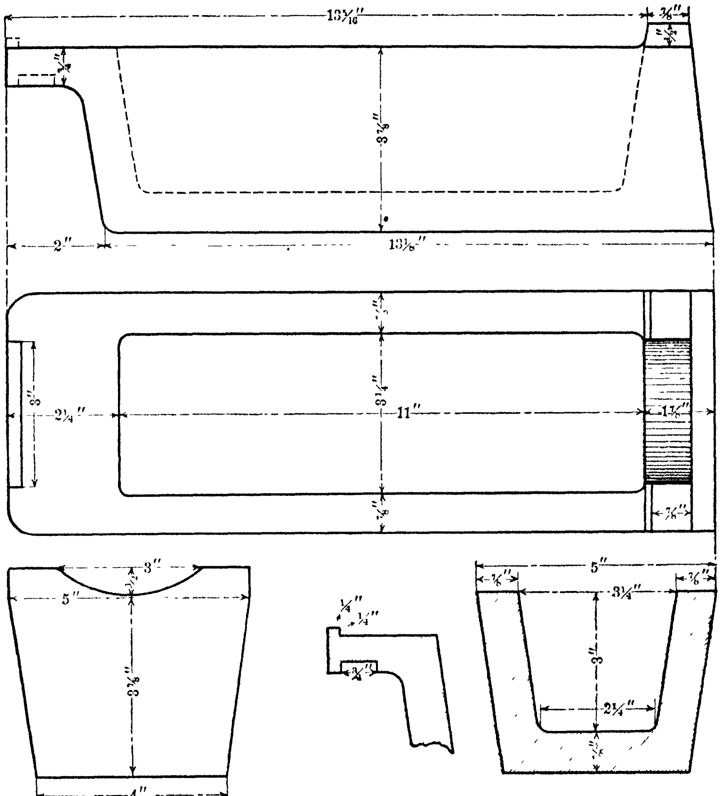
FIG. 4.



iron can be obtained it will give perfect satisfaction. The moulds used by the author endure several thousand operations before becoming worthless. The life of the mould is, of course, shortened by too frequent use while hot, and enough moulds, therefore, should be provided to prevent the temperature from reaching a red heat. The weight of the mould shown in Fig. 5 is about 35 pounds. Although such a weight seems excessive, the advantages of a thick, heavy mould are apparent. The metal cools more quickly, and as the mould does not get as hot as one of lighter weight, its life is consequently prolonged. If properly made, there is little need of lifting the entire mould from the floor when dumping the ingots; raising the end with the tongs and inverting it usually suffices to release the ingot. If the surface of the mould is rough, a slight blow with a hammer is necessary, an operation which can be done with one hand while the other holds the mould.

In Fig. 6 are shown the various forms of ingots occurring in the metal trade. No. 1 is the common form of brass ingot. Its weight averages about 15 pounds. No. 2 is a copper ingot weighing about 15 pounds. The shape of this ingot differs materially from that of the others, the three "heels" being cast so as to furnish a means of cutting the ingot, if sufficiently

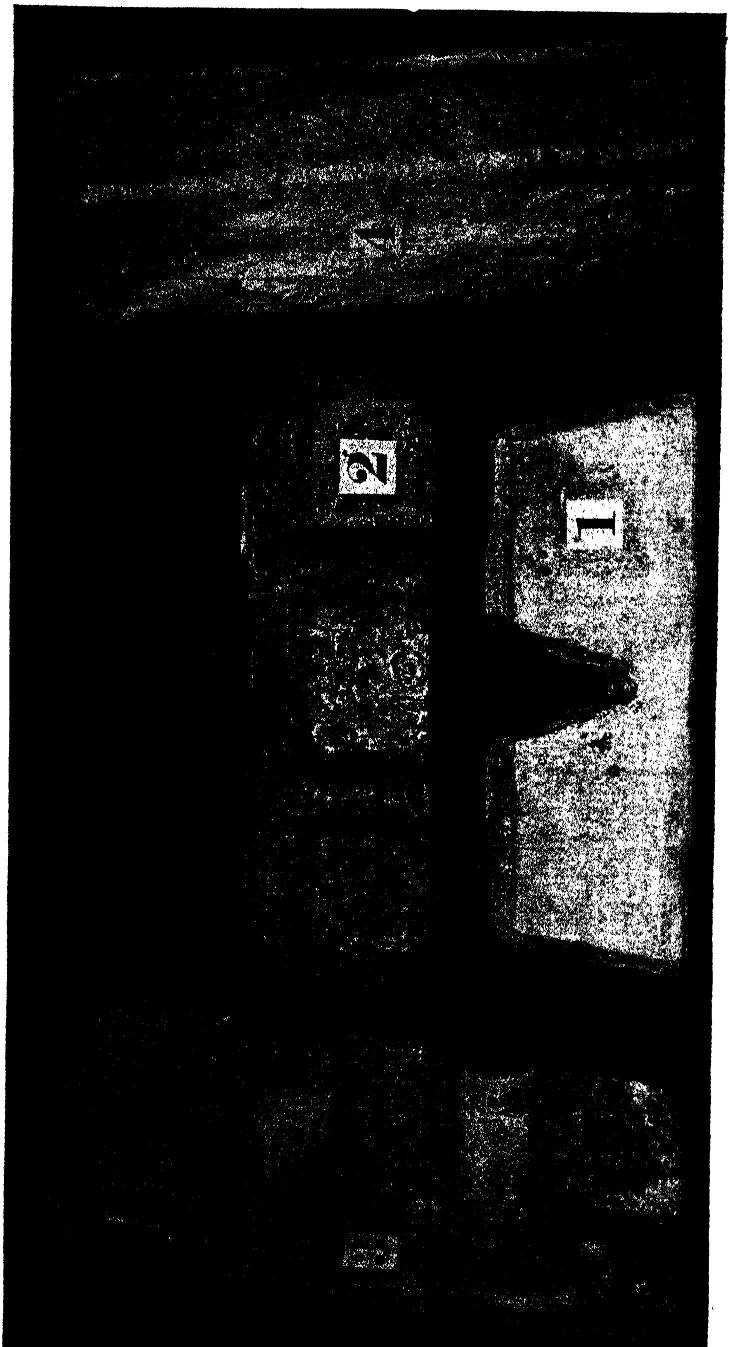
FIG. 5.



INGOT-MOULD FOR BRASS.
Weight, 35 lbs. Scale, 3 in. = 1 ft.

large shears are not available. To accomplish this process the ingot is partially cut through between the "heels" by a hack and sledge, and then broken by one or more heavy blows from the hammer. The ingot shown as No. 1 can be cut in the same manner, but the web between the "heels" is usually so thin that no trouble is experienced in breaking. This form was given to copper ingots before the advent of the power-shear in

FIG 6.



Types of Copper and Brass Ingots.

the brass industry, and has been rigorously followed up to the present time.

No. 3 is an English brass ingot weighing about 23 pounds. In this the depression between the "heels" is so shallow that it would be an extremely difficult matter to cut one of these ingots by hand. Except to preserve the copper type of ingot, such a form is of no advantage.

No. 4 is the form of ingot adopted by the author. The average weight is about 20 pounds. After using such ingots for several years no obstacles have been encountered on account of the shape; in fact they have proved superior for many reasons:

1. The shape of the ingot is suitable for handling with tongs with a minimum danger of slipping. Such a difficulty is often experienced with copper ingots, due to the bottoms of the "heels" being convex.

2. The ingot does not adhere to the mould. With ingots of the shape of Nos. 1 and 2 trouble is often encountered on account of the metal shrinking around the bridge, and thus adhering to the mould when an attempt is made to release it.

3. There is less danger of the metal penetrating the mould. Such a difficulty often occurs, especially with phosphor-metals, when pouring an ingot, through the metal striking the bridge in a continuous stream. In a mould without a bridge the metal does not strike the mould except at the beginning of the pouring, but comes in contact with the metal previously poured.

4. The mould weighs less for a given size of ingot.

In order to avoid the necessity of cutting an ingot to obtain a given weight, the metal can be poured at different levels in the mould so that the size of the ingots may vary enough to furnish any particular weight. In remelting alloys, however, such a proceeding is generally unnecessary.

Notes on the Bertrand-Thiel Process.

BY JOSEPH HARTSHORNE, STOWE, PA.

(Atlantic City Meeting, February, 1898.)

THE attention which this process has attracted, especially in Europe, has led me to believe that members of the Institute would be interested in a report of the progress which has been made in the year and a half that has elapsed since my paper on the subject was read at the Colorado Meeting of this Institute. As already described in that paper (*Trans.*, xxvi., 380), this process is a modification of the open-hearth steel manufacture, whether acid or basic, consisting essentially in the division of the charge between two furnaces, one of which is charged about two hours earlier than the other, and tapped into the other after the charge in the latter has become fluid, the slag of the first bath being removed from it during the transfer.

This description still remains true as far as it goes, but further experience has demonstrated that circumstances will very materially modify the relation of the two furnaces to each other, and the character of the work done in each of them. The function of the primary furnace has become more clearly and entirely that of doing the rough work of desiliconizing and dephosphorizing, while that of the second or finishing-furnace has become almost entirely that of decarburizing and finishing the heat. To indicate these developments and their results is the object of this paper.

In order to avoid needless repetition, I shall try to accomplish part of this object by giving a short *résumé* of what has been published elsewhere on this subject since my former paper was read. This will at least enable those interested to study the matter for themselves from original sources.

Since that time the following papers have been read and published:

"The Bertrand-Thiel Process of Making Steel," by Percy C. Gilchrist (Cleveland Institution of Engineers, Session of 1896–

1897). This paper was widely published in a condensed form in the technical press, the fullest abstract that I have seen being in the *Iron Age* of April 8, 1897.

"Notes on the Practice of the Combined Open-Hearth Process of Messrs. Bertrand and Thiel," by Ernst Bertrand (Iron and Steel Institute, 1897, No. 1, p. 115).

"Der Bertrand-Thiel Process," by Otto Thiel (read before the *Verein Deutscher Eisenhüttenleute*, and published in *Stahl und Eisen*, May 15, 1897, p. 403).

"The Bertrand-Thiel Modification of the Open-Hearth Process," by J. S. Robeson (*Proceedings of the Engineers' Club of Philadelphia*, vol. xiv., p. 167).

Numerous discussions and remarks on these papers and on the process have been published, but I think that the following will cover all that is necessary to a clear understanding of the questions involved:

A criticism from an engineering point of view by B. Dawson, in the *Iron and Coal Trades Review*, vol. liv., p. 166.

Articles in the *American Manufacturer* of February 26th and March 12, 1897.

An article by Joseph Hartshorne in the *Iron Age* of April 15, 1897.

Presidential Address of Mr. F. W. Paul before the West of Scotland Iron and Steel Institute, October, 1897; published in the *Iron and Coal Trades Review*, October 20, 1897.

Mr. B. Talbot has an article in the *Iron Age* of June 17, 1897, which bears on the subject.

Mr. Gilchrist's paper, mentioned first above, is a monument of indefatigable industry and minute analytical investigation. The charges studied by him were principally made up of pig and scrap, although a few had only 10 per cent. of the latter. His main contention is that the great heat developed and the manipulations employed allowed the operator to avail himself, to a greater extent than in the old form of the open-hearth process, of the reducing powers of silicon, phosphorus and carbon, and by this means to reduce sufficient iron from the oxide of iron of the ore added to the bath to more than offset the natural and inevitable loss due to the impurities removed, and, therefore, to obtain a greater yield of iron in the shape of ingots than was originally charged in the shape of pig, scrap and de-

oxidizers or recarburizers. He is also of the opinion that an exceedingly large increase of product per furnace can be obtained by this process in a properly designed plant. Mr. Gilchrist is, of course, solely responsible for the contentions and conclusions set forth in this paper, and I do not find that either of the inventors is so radical or enthusiastic in these respects as he is. While I do not feel able to agree with Mr. Gilchrist that the facts set forth in his paper warrant all of his conclusions, I think that no one can deny that he has shown grounds for the faith that is in him, and has furnished a mine of information which each one can work for himself.

Mr. Bertrand's paper treats of the process in general, but is more especially devoted to the study of charges containing 90 per cent. of pig-iron, or more. The pig contains 3.8 per cent. of carbon, 1.6 of phosphorus, 1.0 of silicon and 1.0 of manganese. Of the 12 heats tabulated, 9 show more iron in the shape of ingots than was charged in the shape of metal, while 3 heats show less iron than was so charged. The greatest loss shown was 2.36 per cent., while the greatest gain was 6.23 per cent. Since the pig-iron contained constituents which would cause a natural or inevitable loss of 7.4 per cent., a loss of 2.36 per cent. means that 5.04 per cent. of iron was gained by reduction from the oxide of iron, while a gain of 6.23 per cent. means that 13.63 per cent. was reduced. This does not, of course, show all the iron that was reduced and saved, since some iron was lost by being entrained in the slag, and in other ways, which must also have been replaced by iron reduced from the oxide.

Of course it is not claimed that this utilization of the reducing powers of silicon, phosphorus and carbon is peculiar to the Bertrand-Thiel process, or that it has not been realized to a certain extent by some of those operating the older forms of the open-hearth process. It is, however, believed that the conditions developed in the Bertrand-Thiel process are such as to allow a much greater gain from this source than has hitherto been generally possible or, at least, achieved. The results obtained at Kladno certainly seem to warrant this belief.

The time-records of the heats reported on by Mr. Bertrand show that with 90 per cent., and over, of pig-iron, the heats were in the primary furnace 4.5 hours and in the finishing-

furnace 2.25 hours; that is, the finishing-furnace was empty half the time. In a properly constructed plant, consisting of two primary furnaces and one finishing-furnace, there should be made between 8 and 10 heats in the 24 hours, proper time being taken for charging and repairs to lining. This means that each furnace would produce 3 heats per day from practically straight pig-iron, which is at least 50 per cent. more than the same furnaces could produce if worked separately.

Mr. Thiel's paper is largely based on the same set of experiments as were those of Messrs. Gilchrist and Bertrand. The heats included in his tables comprise all those given by Mr. Bertrand, 7 of those given by Mr. Gilchrist, and 14 new ones. Mr. Gilchrist includes in his tables 11 heats not given by either of the others. There has been, therefore, a total of 43 separate heats reported in full detail in these three papers. Mr. Thiel has so arranged the material in his tables that they are very convenient for study and for bringing out the salient facts very clearly and forcibly. He also gives very interesting comparisons between the Bertrand-Thiel process and the basic Bessemer and ordinary open-hearth processes.

As it may seem curious that out of so many heats reported there are none which are consecutive in number, it may be well to explain that at Kladno the heats from the open-hearth furnaces and those from the Bessemer converters are cast in the same arrangement of pits, and are numbered consecutively as they are cast, instead of being numbered in one series for the Bessemer and another for the open-hearth.

Mr. Robeson's paper is a general study of the process, which is based on the information given in the papers above mentioned, but does not report any new data or results.

Since the publication of Mr. Bertrand's paper, the only experimental investigation at Kladno has been in the treatment of molten metal taken directly from the blast-furnace. The arrangement of the plant is very inconvenient for such purposes; the available transfer-ladles for the molten metal were only of 5 tons' capacity, and the metal from the blast-furnaces was required for other purposes. For these reasons, only four heats were made in this way. The first two furnished no very definite information as to time, since there was so much delay in getting the metal from the blast-furnaces that the first ladle-

ful was in the furnace for some time before the second was ready to be added to it. They indicated, however, that a very considerable saving in time would be effected.

The history of the third heat is as follows:

Materials Used.

Charge.	Primary Furnace. Kilos.	Finishing-Furnace. Kilos.
Magnetic iron-ore,	2100	1100
Lime,	1000	430
Forge-pig (cold),	3000	
Forge-pig (molten),	9000	

The cold pig-iron was used to complete the filling of the furnace to its full capacity, and was necessitated solely by the small capacity of the molten-metal transfer-ladles. The cold pig, lime and ore were charged together, and when the former began to melt the molten metal was poured upon it.

Time Schedule.

Primary Furnace	P M.
First ladleful charged at	6.
Second " " "	6.15
First test taken at	6.25
Second " " "	7.20
Tapped into finishing-furnace at	7.25

Time in furnace, 1 hour and 25 minutes.

Finishing-Furnace	P M.
Heat received from the primary furnace at	7.25
Slag fluid, and first test taken at	7.50
Second test taken at	8.30
Third " " "	8.55
Heat tapped "	9.15

Time in furnace, 1 hour and 50 minutes.

The total time from beginning to charge the first ladleful into the primary furnace to tapping the heat from the finishing-furnace was 3 hours and 15 minutes.

Analyses.

	C. Per cent.	Si. Per cent.	P. Per cent.	Mn. Per cent.
Pig-iron,	......	1.00	1.40	0.50
Primary furnace, 1st test,	3.84	......	1.17	.....
" " 2d "	2.12	......	0.36	......
Finishing- " 1st "	1.00	......	0.01	......
" " 2d "	0.45	......	0.01	......
" " 3d "	0.153	......	0.01	......
Finished steel,	0.063	......	0.01	0.151

Fifty kilos (0.42 per cent.) of 80-per-cent. ferro-manganese was added for deoxidizing. The ultimate strength of the steel was 48,832 pounds per square inch and the elongation was 34 per cent. in 8 inches. The loss on the heat was 0.7 per cent.; that is, 99.3 per cent. of ingots was obtained.

The production of a plant running on such a basis depends entirely on the rate and capacity of the finishing-furnace. Only the first heat at the beginning of a campaign, such as starting up at the beginning of the week, will require the total time in the two furnaces, since the finishing-furnace will then have to remain empty until it receives the metal from the primary furnace. On the next heat, and on those following, the two furnaces will be working simultaneously, and the first half of one heat will be treated in the primary furnace while the last half of the previous heat is being treated in the finishing-furnace. In the heat above reported, the time in the finishing-furnace was a little less than two hours. If one hour be allowed for repairing the furnaces, there would be one heat every three hours, or eight in the 24 hours. In view of the short time the heats remain in the furnaces and the consequent small repairs that will be necessary, half an hour should be ample allowance for this purpose. In that case there should be a heat every two hours and a half, or about ten in the 24 hours.

The record of this heat, therefore, indicates that from eight to ten heats can be made per day from molten blast-furnace metal, allowing from a half an hour to one hour intermission between heats. This means four or five heats per day per furnace, since it is evident that the proper combination for molten blast-furnace metal of such composition will be two furnaces, the time the metal remains in each furnace being so nearly the same.

The fourth heat was made from the same kind and amount of molten metal and with the same amount of ore and lime. It was held in the primary furnace until the carbon was run down much lower, in order to see if any advantage would be obtained thereby. The result showed that it is better to tap out of the primary furnace as soon as the phosphorus is properly reduced. The time in the primary furnace was 2 hours and 55 minutes, and in the lower furnace 2 hours and 10 minutes. No time was gained, therefore, from the extra hour in

the primary furnace. The finished metal in this heat contained 0.06 per cent. carbon, 0.04 phosphorus and 0.14 manganese.

With the exception of the more or less experimental heats just described and those reported in the papers above mentioned, the regular routine work has been steadily continued at Kladno. The results have been very satisfactory, considering the imperfections of the plant, which, as explained in my former paper, is a fortuitous make-shift. The charges have been from 21 to 22 tons, 12 to 13 tons of which are charged in the primary and the balance in the finishing-furnace. These charges have, as a rule, consisted of about 60 per cent. of pig-iron and 40 of steel scrap. The pig contains about 2.5 per cent. of phosphorus and 0.5 of silicon. The production has been 29 to 30 heats per week, averaging a little over 100 tons per day or 600 per week. The finishing-furnace has generally been empty half the time. Owing to the internal economy of the company and its allied interests, it is sometimes better policy to devote as much as possible of the scrap to other uses than to charge it into their own open-hearth furnaces. Under these circumstances they adopt the following practice: The upper furnace is charged with 16 to 17 tons of forge-pig containing 1.5 per cent. of phosphorus and 0.8 to 1.0 of silicon. The charge of the lower furnace consists of 4 tons of scrap and 2 tons of Styrian pig, which is low in phosphorus. Working in this manner they get from 28 to 30 heats per week, although the charge contains about 82.5 per cent. of pig-iron.

In working with cold pig, it has been found to be more advantageous to add the ore to the bath in small portions (say 200 kilos) at a time, rather than to charge it all at once at the beginning. By this means it is possible to reduce the phosphorus to a low limit without materially affecting the carbon. The metal now tapped from the primary furnace contains 0.10 per cent., or less, of phosphorus and 2.0 to 2.5 of carbon. It is not unusual to tap metal from the primary furnace containing only 0.06 per cent. of phosphorus.

The process has now been worked out in detail at Kladno for pig and scrap, for cold pig-iron alone, and for molten metal from the blast-furnace. The general results of these investigations may be summarized as follows:

1. *Pig-Iron Alone.*—The plant should consist of two primary furnaces and one finishing-furnace. The finishing-furnace should be at least one-third larger than either of the primary furnaces. From 80 to 90 per cent. of the charge may be put in the primary furnace and the balance in the finishing-furnace. The product will be 8 to 9 heats per day, of a weight depending on the size of the finishing-furnace.

2. *Pig and Scrap.*—The plant should consist of two primary furnaces and one finishing-furnace. The finishing-furnace should be about twice as large as the primary furnace. The charge should be divided between the primary and finishing-furnaces in about the proportion of three-fifths in the former and two-fifths in the latter. Any proportion of pig-iron can be used, up to 60 per cent. at least, without reducing the rate of production. The product will be from 8 to 10 heats per day, of a weight determined by the size of the finishing-furnace.

3. *Molten Metal Direct from the Blast-Furnace.*—The plant should consist of one primary and one finishing-furnace. The latter need not be more than one-third larger than the former, and furnaces of the same size will answer. From 80 to 90 per cent. of the charge should be put into the primary furnace in the shape of molten blast-furnace metal, and the balance into the finishing-furnace. This latter portion can be charged cold, and may be either pig, or scrap, or both, as may be most convenient. The product will be 8 to 10 heats per day, of a weight determined by the size of the finishing-furnace.

4. *Molten Metal and Scrap.*—In case the amount of scrap which is to be disposed of, or is economically available, amounts to more than, say, 30 per cent., the plant should consist of one primary and two finishing-furnaces. This is on account of the time required to melt the scrap in the finishing-furnace. The liquid metal and a little scrap should be charged into the primary furnace, and the balance of the scrap should be charged into the finishing-furnace. If necessary to prevent excessive oxidation of this scrap, a little pig-iron can be charged with it, or carbon can be added in the shape of coke-dust. The product will be 10 to 11 heats per day, of a weight determined by the size of the finishing-furnace.

These modifications of the plant and process cover most

of the cases which will arise in practice. It is possible, however, that the treatment of a pig-iron which is at the same time very high (say 2.5 per cent.) in silicon and very high (say 2.5 per cent.) in phosphorus, may require further modifications of plant and practice. The treatment of such metal is described by Mr. Thiel under the caption of Case II. (*Stahl und Eisen*, May 15, 1897, p. 409), in the light of such experience as had been acquired up to the time his paper was written. This variation of the process will be thoroughly investigated as soon as an opportunity can be made. I may say, however, that, in my judgment, the present indications are that the intermediate furnace will be found to be a needless complication, and that such material can be treated as well in a series of two furnaces as that which was treated in the other cases described.

The economies in fuel, in refractories of both kinds and in basic additions which were noticed in my former paper, have been maintained at Kladno; but these economies, although considerable, are of minor importance compared to the saving due to the greatly increased product which can be obtained from a plant of a given first cost, or, conversely, the greatly decreased cost of a plant for a given production.

Outside of the economic features of the process, upon which stress has been laid, there are other advantages which are due to the peculiar conditions produced. Owing to the treatment in the primary furnace, the metal, as it comes into the finishing-furnace, has in it no silicon, less than 0.10 per cent. of phosphorus, and from 2 to 2.5 per cent. of carbon. It will be at once perceived that this metal is a most excellent one for treating in an acid furnace, when the phosphorus is not to be lower than 0.06 per cent. In fact, under such circumstances, the heat might just as well be finished in an acid furnace. There would be no particular advantage in doing this, however, and it would have the disadvantage of requiring the two processes to be worked at the same time, and thereby reducing the simplicity of procedure. Although the finishing-furnace is lined with basic material and basic additions are made in it, nevertheless the principal disadvantages of the basic process are avoided in using it. The worst of these disadvantages is the heavy body of slag containing large amounts of silica and phosphoric acid, which is especially troublesome as the heat

approaches its finish. Since there is no silicon and but little phosphorus in the metal which is tapped from the primary into the finishing-furnace, very little lime is required as a purifying agent in the latter, and very little slag is made. In fact, one of the principal functions of the slag will be to protect the metal from too vigorous oxidation by the flame. Owing to this thin body of slag, the flame will perform its double function of heating the metal and oxidizing the carbon in a much more rapid and effective manner than in the ordinary furnace. The heat will be ready to tap, as far as temperature is concerned, very soon after it reaches the finishing-furnace, and the phosphorus will also be very rapidly reduced to the lowest limits. The rapid elimination of the phosphorus and carbon is also aided by the fact that the oxide of iron in the slag is very little diluted by impurities, and its action is more direct and effective. As an instance of the rapid elimination of the phosphorus, the heat described above may be cited. The metal was tapped from the primary furnace at 7.25 P.M. Five minutes before it was tapped a test showed that the metal contained 0.36 per cent. of phosphorus. A test taken from the finishing-furnace at 7.50 P.M., 25 minutes after tapping began, showed 0.01 per cent. of phosphorus; that is, the phosphorus dropped 35 points in 30 minutes, during which time the transfer from one furnace to the other took place.

Since there is nothing to be removed from the metal in the finishing-furnace but carbon, the operator can give his undivided attention to that alone. This enables him to work much more carefully and exactly. It also makes it possible to produce high-carbon steel, say, of from 0.25 or 0.30 to 1.0 per cent. and upwards of carbon, by arresting the refining when the carbon in the metal has been brought down to the desired amount, and to do this with even greater ease, regularity and certainty than is now possible in the acid process. Two instances of this practice were cited by Mr. Thiel in his paper above referred to (*Stahl und Eisen*, May 15, 1897, p. 413). He there says:

"A test of No. 87,110, taken from the lower furnace after three-quarters of an hour, had the following analysis: 0.732 carbon, 0.022 phosphorus and 0.086 manganese; and one taken from No. 87,137 after the same time had 0.767 carbon, 0.07 phosphorus and 0.C96 manganese. There were 20 kilos of ground coke

added in the ladle to the first heat and 30 kilos to the second one, since it was feared that the refining had been carried on too vigorously. The steel had an ultimate strength of 80 kilos (113,859 pounds per square inch). The steel rolled perfectly."

By consulting Table II. of Mr. Thiel's paper, it is seen that in both cases 0.5 ton (0.42 per cent.) of ferro-manganese was added to deoxidize. The finished steel of heat No. 87,110 contained 0.768 per cent. carbon, 0.041 phosphorus and 0.357 manganese. The finished steel of No. 87,137 contained 0.624 per cent. carbon, 0.035 phosphorus and 0.195 manganese.

The deoxidizers and the recarburizers, when any are used, can be added in the furnace with the same ease and certainty as is now the case in the acid process, since the oxide of iron in the slag can be closely regulated in amount and prevented from having too vigorous action, and there are no reducible impurities present in sufficient amount to introduce other uncertainties.

Finally, I may say that the process has begun to spread on the Continent. In a letter recently received from Mr. Thiel, he tells me that Schneider et Cie., of Creusot, France, have carried out an extended series of experiments which gave very satisfactory results, and, in consequence thereof, have taken a license to use the process, and are building a new open-hearth plant for this purpose, which they expect to start in operation about June 1st of this year. This license was granted by the *Eisenhütten-Actien-Verein-Düdelingen*, of Düdelingen, Luxemburg, to which the French and Belgian patents have been sold. The Düdelingen company have also taken licenses for their own works in Luxemburg, as well as for the works of Metz et Cie.

Notes of a Reconnaissance from Springfield, Mo., Into Arkansas.

BY E. J. SCHMITZ, NEW YORK CITY.

(Atlantic City Meeting, February, 1898.)

THIS trip, beginning at Springfield, August 12, 1897, and ending at the same place fourteen days later, extended south and southeast through Christian, Ozark and Taney counties,

Mo., and into Marion and Boone counties, Ark. Its object was a reconnaissance of the zinc, lead and other resources of the counties of Northern Arkansas named, and also a rough and rapid view of the topography, etc., of the country traversed, as basis for an opinion concerning the practicability of a projected railway.

Altitudes were estimated from barometer-readings, that of Springfield, Mo. (1430 feet), being taken as a base. On my return, the barometer showed 1370 feet altitude at Springfield.

On the first day's journey, which ended at Taneyville, Mo., about 40 miles south of Springfield, we passed Ozark (about 15 miles from Springfield; altitude about 1250 feet), traveling over a rolling country, showing an exposed face, 30 feet thick, of conchoidal limestone, a little slate, and (at Ozark) flat-bedded quartzites. This quartzite is honeycombed, and occasionally somewhat conglomeritic. Bull Creek (11 miles from Ozark; altitude about 1200 feet) is reached after passing a considerable hill, 1600 feet high. Here the prevailing rock was chert, with some cherty limestone and some sandstone, a face of the latter being observed, 100 feet thick. About 14 miles from Ozark, Pine Mountain was passed, and Taneyville (altitude 1300 feet) was reached a dozen miles beyond.

The whole geological formation thus far observed is flat-bedded, and undoubtedly belongs to the Lower Silurian. More precisely, I consider it to be Potsdam and Calciferous (Knox sandstone). The limestones mentioned must belong to the Knox dolomites.

Continuing in the same general S.S.E. direction, we passed on the next day Beaver, Cedar and Elbow creeks, in the order named (altitude of each about 800 feet) and the State line between Missouri and Arkansas at White river (about 21 miles from Taneyville; altitude about 620 feet); and passing through the town of Lead Hill (altitude about 700 feet) we reached Sugar Loaf creek (about 16 miles from White river; altitude about 1100 feet).

About 2.5 miles N.W., on the Wild Cat branch of Sugar Loaf creek, is the Swansea mining property. Here we found a hill 330 feet high, sloping steeply to the branch, and divided by a gap 250 to 300 feet wide. In the gap, only quartzite

could be seen, but on both sides there were visible two clearly defined horizontal beds, one of red marble, about 30 feet thick, lying about 110 feet above the branch, and the other of limestone and quartzite, 20 feet thick, lying 100 feet lower. Of this 20 feet the upper half is flaggy siliceous limestone, while the lower half is whitish, honeycombed and somewhat conglomeritic quartzite, containing a small amount of zinc. (A sample analyzed showed 1.36 per cent. of zinc.) The two beds thus exposed on both sides the gap might be supposed to have been continuous across that space, but no trace of bedding was observed in the quartzite of the gap. The lower bed, though occasionally interrupted, is generally persistent, and holds with considerable regularity its position relative to the red marble. The latter, being almost everywhere observable, presents a convenient horizon to serve as base and landmark for observations of ore-bearing beds. It has no commercial value.

The zinc-ore in the honeycombed quartzite is finely disseminated blende and carbonate. The upper half of the bed (siliceous limestone) also shows occasionally flat deposits or fillings of zinc-blende.

About 60 feet below the lower bed, and in the side of the gap, the Swansea shaft has been sunk. To the depth of 23 feet (which was all that could be seen, the rest being under water) it shows quartzite and some conglomeritic siliceous rock. About one-half of the exposed faces is clay. The quartzite appears in fragments and boulders. About two-thirds of the rock taken out seems to be more or less ore-bearing.

Three other outcrops of quartzite "ledges" were observed in the vicinity. The one already described reappears at intervals along the face of the hill. Of the others, one is apparently not a horizontal bed, but a fissure or "blow-out."

The Frisco mine, near by (N.W. ¼ of S.E. ¼, Sec. 17, T. 19 N., R. 18 W.), shows the main ore-bearing quartzite about 80 feet above a branch of Sugar Orchard, in a heavy bluff of siliceous strata, partly sandstone, partly conglomeritic quartzite. The upper half of the bed, which is at about the same level as at the Swansea mine, has here become sandstone or quartzite; but some bars of siliceous limestone can still be found. The ore is mostly zinc-blende, disseminated in the quartzite or siliceous rock. Carbonates occur, but very variably. The quartzite

which carries most ore is honeycombed. The main bed or ledge is exposed for 90 feet in length, and has been worked for a thickness of 10 or 12 feet. I estimate that the whole ledge, as exposed in the face of the Frisco bluff, carries about 30 per cent. of workable ore, mostly blende. Analyses of samples of such ore, representing the whole face of 90 by 12 or 1080 square feet, showed 32.16 and 31.31 per cent. of zinc respectively. A cubic foot of the ledge weighs from 200 to 220 pounds. Taking the smaller figure, and assuming 30 per cent. of ore in the rock and 30 per cent. of zinc in the ore, we should have for each 10 feet from the outcrop, on the length and thickness given,

$$\frac{90 \times 12 \times 10 \times 200 \times 30}{2000 \times 100} = 324 \text{ tons of ore,}$$

or

$$\frac{324 \times 30}{100} = 97.2 \text{ tons of zinc.}$$

The Frisco main ledge is reported to be traceable also on the opposite hill to the east.

Continuing north along the creek, we found during the next two or three days many openings in which the zinc-horizon was exposed—mostly in its usual level position, but occasionally locally tilted. The most important development observed was MacGregor's lead mine (N.W. ¼ of Sec. 26, T. 20 N., R. 17 W.), visited August 17th. Here the bluft rises from 830 feet, the altitude of the creek, to 1110 feet. At 1010 feet the red marble appears, and about 75 feet lower down (the interval being occupied with limestone and quartzose limestone) is a formation 11 to 12 feet thick, composed of (1) quartz, 3 feet; (2) siliceous shale, 2 feet; and (3) shale, etc., 6 feet. In (2) and (3) occur lead carbonate and galena. The formation is flat.

On August 18th we proceeded to Yellville, *via* the top of Lee mountain, the Bear Hill property, and the New York Zinc and Lead Company's property (all showing outcrops of the stratified zinc-bearing bed). The next day we pursued our examinations southward to and along Rush creek, to the Morning Star mine, situated in Sec. 10, T. 18 N., R. 15 W., etc. Here the altitude of Rush creek was estimated to be 540 feet, and the top of the bluff 950 feet. The outcrop of the red marble

occurs at 910 feet, and the zinc-horizon is 100 feet lower down, quartzite intervening. There is also an important opening at an altitude of 640 feet, where a large amount of ore is exposed. The developments on the ore-bearing bed are distributed over more than 500 feet of its length, but not connected with each other.

Opening No. 1 enters the mountain nearly 80 feet, with a length of about 30 feet. It shows quartzite and slate, and about one-third of the surface exposed is ore (carbonate).

No. 2, a small opening, likewise shows ore and rock mixed —about 40 per cent. of the former—both carbonate and blende. It is 50 feet from No. 1. In the next 360 feet there are four subordinate openings, showing ore and rock mixed. Then comes No. 3, from which about 120,000 cubic feet of material has been already removed, a large percentage being zinc-ore. The walls of the working now show about 50 per cent. of ore. An opening higher up has furnished about 10,000 cubic feet, and its faces show from 40 to 50 per cent. of ore.

No. 4, driven about 90 feet, shows about 33 per cent. of ore, principally carbonate. No. 5, a cut and tunnel, the latter driven in about 75 feet (altitude 600 feet), shows less ore—not over 20, probably not over 10 per cent. of the whole mass removed. No. 6 has about 25 per cent. of carbonate ore. All the ore on this property is above water-level, and there is evidently a large amount of it, imperfectly exposed by present developments.

Opposite the Morning Star is a large fault in the formation, with a throw of 300 feet. This is a rare occurrence in this field.

On August 20th we finished our examination of the Morning Star, and inspected the Mackintosh, the Silver Hollow and the Red Cloud property; the two latter being on the east side of Buffalo river.

On our return from this point to Springfield, several outcrops and developed exposures of ore were examined. Among them were large outcrops of carbonate (with but little blende) at the Big Buffalo mine, Sec. 5, T. 19 N., R. 16 W. The property has four openings. The Buffalo Bill mine, half a mile further north, shows ore in quartzite. Various points between this and Lee's mountain, and along the mountain, showed carbonate outcrops.

The same was true of points on the road followed August 24th, 25th and 26th, via. Isabell in Ozark county, Mo., Thornfield, Rome, Roy, Arden's, and Finley creek, Henderson and James river to Springfield. Two zinc- and lead-mines, viz., Bryer's and Thomas Carsener's mine, located on Turkey creek, 7 or 8 miles S.W. of Springfield, about 1 mile north of James river, and on the Kansas City, Memphis and Gulf Railroad, are reported to be paying.

General Observations.—The rocks of the region traversed in this reconnaissance extend from the Potsdam up to and including the Quebec group. I estimate their thickness here represented at over 1000 feet.

From lowest valleys to highest ridges the elevation ranges between 400 and 1600 feet above tide, but the ridges are not commonly more than 400 to 600 feet high. The country is rugged, but contains some large valleys, and pretty good agricultural lands. Good second- and third-class timber is fairly abundant. There are numerous springs and streams, but only the White river and its Buffalo fork are navigable—and these only during five or six months of winter, late autumn and early spring.

The roads are hilly and rough, and the lack of transportation-facilities has greatly retarded the development of the mineral resources. Of these, the zinc-ores were first discovered 20 or 25 years ago, yet have never been worked on a commercial scale. Railroad-schemes have been repeatedly projected for this region, and such a scheme is now said to be in contemplation. The cost will be considerably higher than that of the average railroads on the prairies. It will probably amount to between $15,000 and $20,000 per mile. But such a road, if constructed, would undoubtedly stimulate an active development of the mineral resources of the region.

The most valuable of these appears to be the zinc-deposits, the occurrence of galena being, so far as is now known, subordinate. The zinc-ores occur in three forms of deposit: (1) impregnating stratified beds; (2) filling irregular pockets (usually near such beds); and (3) in fissures. The latter I do not think likely to prove deep and largely productive. In my judgment it is the first class of deposits upon which the future of the industry must depend. The indications are that several horizons bearing zinc-ore in quartzite and siliceous rocks ex-

tend over a large area. Only actual further development can show to what extent they are everywhere workable. But I am satisfied, from what I have seen of them, that they will furnish a very large tonnage when properly opened and worked. The ores are of pretty high grade, as the table of analyses below demonstrates.

Good coal is reported 15 to 40 miles S. and S.W. of Rush creek. The coal-measures of the great southwestern field join the Silurian district of Arkansas and Missouri on the south, and there is every reason to expect good coking and other coal from that source, which would be important for the reduction of ores in the mining district itself.

Table of Analyses of Samples.

Analysts, Endemann & Saarbach, New York City.

No. of Sample.	Zinc. Per cent.	No. of Sample.	Zinc. Per cent.
1	1.36	13	32.61
2	30.49	14	31.31
3	2.17	15	17.53
4	35.60	16	22.01
5	1.09	17	19.02
6	37.37	18	0.82
7	15.76	19	16.85
8	0.78	20	0.54
9	42.61	21	45.00
10	37.91	22	39.78
11	51.09	23	25.27
12	26.36	24	35.05

1. Yellow outcrop, honeycombed. 2. Blende (black jack), good sample from Morning Star mine. 3. Poor sample of carbonate in quartzite on Rush creek. 4. Good sample of carbonate, etc., in calcite from Mill creek. 5. From S. 1, T. 19, R. 18 W. Two pieces of quartzite above shaft, considered to be from 30 feet in place. 6. Ore from S. 17, T. 19, R. 18 W. 7. Matrix on lean ore from shaft, S. 17, T. 19, R. 18 W. 8. Quartzite float, high up on hill, S. 17, T. 19, R. 18 W. 9. Ore from S. 17, T. 19, R. 18 W. 10. Ore and quartzite from No. 2, S. 17, T. 19, R. 18 W. 11. Ore from S. 17, T. 19, R. 18 W., Shaft. 12. Matrix quartzite from No. 2, S. 17, T. 19, R. 18 W. 13. Frisco mine. 14. Ore from shaft No. 1, S. 17, T. 19, R. 18 W. 15. Matrix from ore in shaft, S. 17, T. 19, R. 18 W. 16. Average of ore and matrix from quarry No. 1, examined August 23, 1897. 17. Average from quarry No. 2, examined August 23, 1897. 18. Outcrop on left side of Frisco Mt., going up from Mill creek. 19. Ore showing 5 blende crystals from Rhodes hill on Mill creek (right hand), S. 16, T. 19, R. 18 W. 20. Sample from outcrop on Boine. 21. From top of Rhodes hill at head of Mill creek. Highest ore outcrop, altitude 1250 feet, S. 4, T. 19, R. 18 W. 22. Same place as large block, S. 16, T. 19, R. 18 W. 23. Poor ore in matrix on N. side of dry hollow branch of Mill creek, S. 17, T. 19, R. 18 W. 24. About ½ mile E.N.E. from Frisco, on same side (W.), S. 18, T. 19, R. 18 W.

Scorification and Cupellation Without Muffle.—A New Furnace and Method for Gold and Silver Assays.

BY GEORGE A. KOENIG, PH.D., MICHIGAN COLLEGE OF MINES, HOUGHTON, MICH.

(Atlantic City Meeting, February, 1898.)

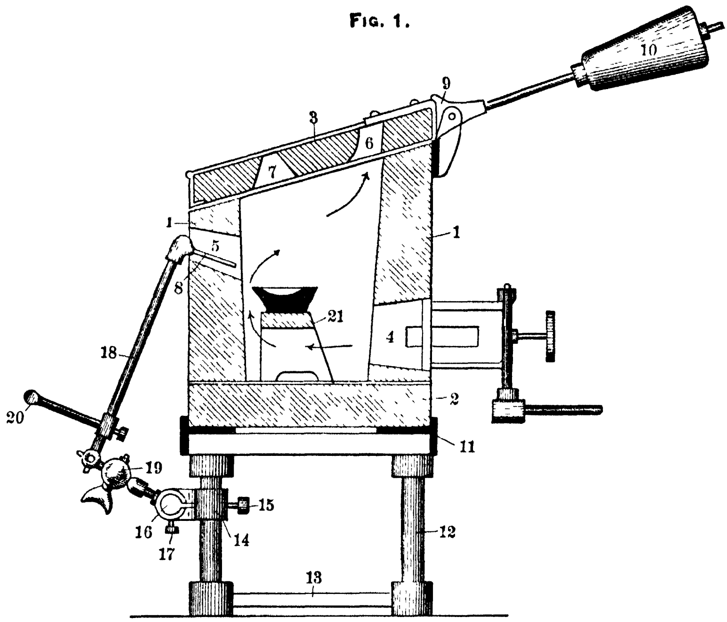
THIS new departure in assaying is the outcome of a long-felt desire to shorten the time required in muffle-assaying, as well as to do both crucible- and scorification-work in one furnace. The first object in view was the gold-assay, which must always be a combination of the crucible- and muffle-work. The large lead buttons consume much time in their reduction by scorifying and cupelling. The Hoskins gasoline furnaces have been a great convenience, for years past, in that class of practice, where only one assay was required at a time. But even with them I found myself hampered, since the crucible-furnace only takes one crucible at a time, and my practice implies always duplicate assays. Another difficulty was to get the muffle up to the necessary heat and to maintain it at that heat. Whenever I had pumped myself into perspiration and out of breath, the question presented itself, "Is the muffle necessary to do this work properly?" Hartmann and Plattner and all their blow-piping disciples have shown that very good results are obtainable by playing a direct flame upon the work-lead. Could this be done on the regular muffle-assay scale? I placed a cupel upon a fire-brick, surrounded it with charcoal, and then played upon it with a Bunsen gas blow-pipe. The result was encouraging. With some care a 2.75-inch scorifier charged with 0.1 A. T. of a 100-ounce ore could be melted down and scorified without any apparent loss. I thought then of constructing a furnace with four blow-pipes in a row; but the combination of gas and charcoal seemed unpractical. Instead, I placed an F crucible, bottom up, in a Hoskins crucible-furnace. Upon this support I placed a charged scorifier. The rapidity with which a very refractory charge melted and scorified was simply amazing to me. Then a cupel was put in the place of the scorifier, the lid was moved sideways, and a blow-pipe was

adjusted, just blowing air, since the heat was supplied by gasoline. A 30-gramme lead button disappeared in three and one-half minutes, the lead oxide volatilizing instead of soaking into the bone-ash. I was naturally still more amazed than before. All right so far. But did not the silver also go into the chimney? No; to my still greater amazement, the loss in silver was not much above that of the well-known cupel-draught in the muffle. The preliminary experiments seemed to prove the muffle unnecessary; they were sufficiently encouraging to warrant further expenditure of time and thought. In order to come up to my wants, a furnace must permit the making of a number of assays at a time, if necessary, so as to utilize to the utmost the time and working-capacity of the assayer. It must, furthermore, permit the simultaneous execution of crucible-fusion, scorifying and cupelling. It must allow roasting, and it must work with either gasoline or gas.

The Furnace.—The furnace designed to satisfy these conditions, according to the experience gained, has a rectangular ground-plan 10 by 22 inches in outside measure, and 5 by 17.5 inches inside measure at the bottom. These dimensions accommodate in one operation six F Battersea crucibles, or six 2.75-inch scorifiers, or six cupels. Fig. 1 is a vertical cross-section, showing the furnace in the act of scorifying. At 1—1 we see the front and rear walls, tapering from 2.5 inches at the bottom, to 2 inches at the top. The rear wall is 10 inches high outside; the front wall only 7.25 inches. Thus the lid (3) has a considerable slant backwards, with a vent-slot at (6). The reason for this slant is this. The flame entering through the rear wall at (4) follows the line of the arrows to the vent. If the lid lay in horizontal position the current of the gases would rebound against it and cause disturbing whirls, whereas now the gases only rebound against the front wall and then expand naturally, yielding their heat more fully. There is neither tendency in the gases to escape through the observation-hole at (7) nor through the blast-hole at (8). Much heat is wasted through unnecessary contraction of the flames. When the flow of gasoline is properly regulated there is neither pressure nor suction at the blast-hole; so that neither scorifier nor cupel, though standing before the opening, is needlessly exposed to chill. In roasting at a low red heat there is a natural draught

from the blast-hole to the vent, which would not be equally advantageous with a horizontal lid. For reasons of easy manipulation, the top edge of the front wall should not be much above the scorifiers and cupels, and this constitutes a further reason for raising the back wall, to avoid a crowding of the flame. It is seen that the lids are hinged at (9), and that the hinge-plate is supported upon side-brackets, quite independent of the back wall of the furnace. An adjustable counterweight (10) is mounted on each lid in such a manner that a very slight force will either lift the lid or depress it or keep it opened at

FIG. 1.



Vertical-Cross-Section of Furnace (scorifying).

any desired angle. The four walls form in the present stage of development one piece of non-conducting refractory casing; but a new pattern is contemplated in which the walls will each form a separate, exchangeable piece, they being simply held together by means of two iron bands. An entire sheet-iron outer casing has been tried, but not found of any special advantage. The casing rests upon a 2-inch bottom tile of similar composition as the casing. The bottom plate rests upon a cast-iron frame (11) which is supported by four pipe-legs (12) 7 inches high. The feet of the legs are secured to a double

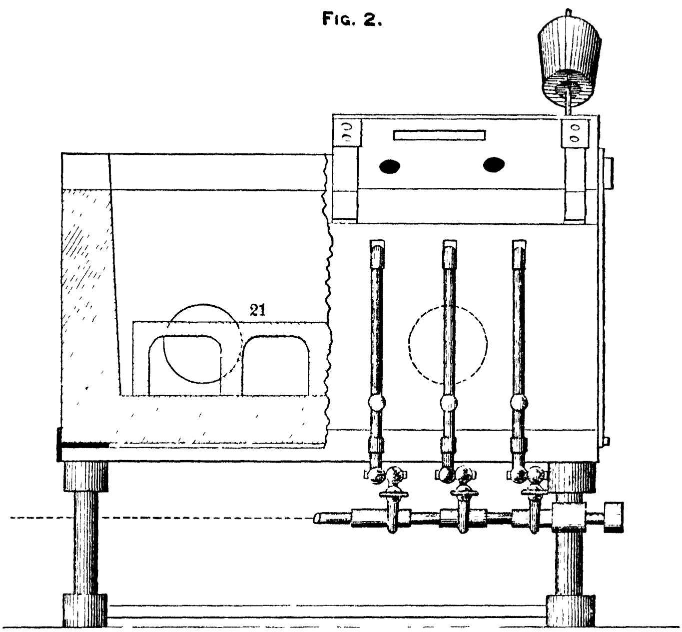
forked brace (13), thus making a simple and rigid understructure. Upon the front legs slide two arms (14) with set-screws (15). The arms carry a main blast-pipe (16) which can be secured in any desirable position by set-screws (17). From this main six branches (18) lead to the blast-nozzles. Each branch is furnished with a double swivel-joint stop-cock (19) and a cast-iron handle (20). For the suggestion and design of this iron understructure I am indebted to my friend and colleague, Professor Kidwell. My first furnaces had shorter legs, cast to the bottom frame. The blast-main was in front of the bottom tile, bringing the stop-cocks midway up the front wall. With such an arrangement the joints came under the influence of radiating heat, expanded and became loose, unless they were screwed up so tightly that in the cold state they would not move freely.

Each branch is entirely independent of the others, and may be thrown fore and aft without moving its neighbors. The swivel-joints allow any horizontal or vertical adjustment of the nozzle with the greatest ease by means of the handle. Since a change is often desirable in the angle at which the air-current strikes the surface of the metal in scorifying or cupelling, such change of angle can be effected by lowering or raising the arm-bracket (14) along the leg. In the figure this angle is large; the knee at (19) is nearly a right angle. By lowering the arm the knee may be changed to a straight line; the air-jet will then be nearly horizontal, a position which is required in roasting, to avoid blowing away ore-particles. At (21) stands the bench or bridge, which supports scorifiers, roasting-dishes or cupels. It is 8 inches long, 2 inches wide at top, 3 inches at bottom, and 3.5 inches high. Any sand crucible, ground to the proper height and placed upside down, could be used as a support in an emergency. In its present form the bench is supported by three legs. The first ones had only two legs. These latter sagged when, after an upset or spill, the scraper was rubbed back and forth over them. The three-legged type wears better.

It has another advantage, in affording a fire-brick mass without interfering with the passing of the fire-gases. The fire-brick mass acts like a Siemens heat-storer. When the furnace is used for crucible-fusions this bench is taken out, and when, after the crucible-fusions, cupelling or scorifying is to be done,

the bench is simply dropped into place by means of suitable tongs. The clear length of the furnace at the bottom being 18 inches, there is ample room for two benches and the diaphragm (22), Figs. 4 and 5. This diaphragm is a fire-tile, 0.75 inch thick, just fitting loosely the cross-section of the furnace. I consider it a very important part of the furnace, because it makes either two independent furnaces or one single large furnace. In a very busy office the diaphragm will be discarded, most properly; the whole furnace being used for a time as a roaster, then as a smelter, and lastly as a cupeller. It is for

FIG. 2.

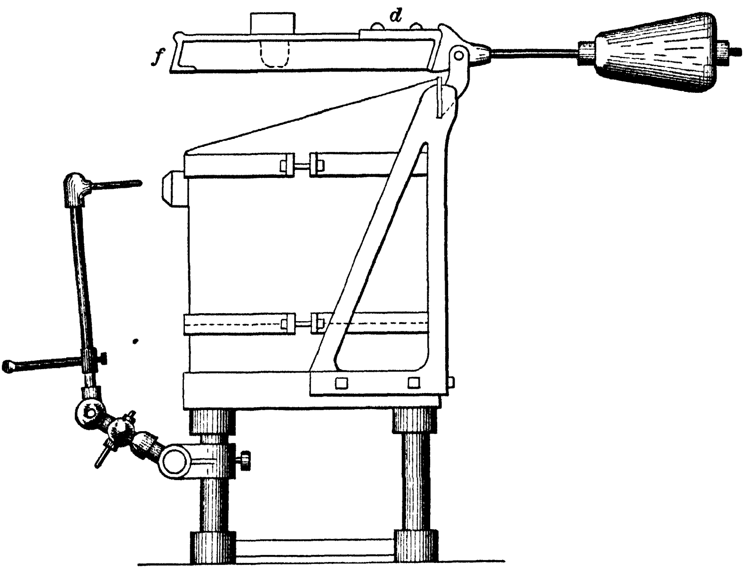


Vertical Half-Section and Front Half-Elevation.

one, two, or three assays at a time, that it comes in as an economizer both in time and fuel. In Fig. 4 the bench has not been placed in proper position; it should be reversed, so as to bring the edge, *a b*, towards the front or blast-side of the furnace. The figures are readily understood. In Fig. 2 one-half of the furnace is seen as front elevation, showing the position of blast-pipes, the tuyeres and the lid. The second half is the vertical section through the bench, without the lid, and with the hole for the burner behind the bench. The center of the latter is located so deep that the flame will not strike against

the top of the bench, but only against its central leg, and will then pass sidewise against the front wall and up in the direction of the arrows. In Fig. 3 is given the end-elevation of the furnace, showing the bracket which supports the hinge-plate, and the plugs which close the tuyeres (when the blast is not required), as well as the two observation-holes in the lid. The position of the latter indicates its balanced condition. Notice here also the two bands of stove-pipe iron which brace the casing. Since the drawing was made, I have found that one single band, only one-half inch wide, is quite sufficient to hold

FIG. 3.

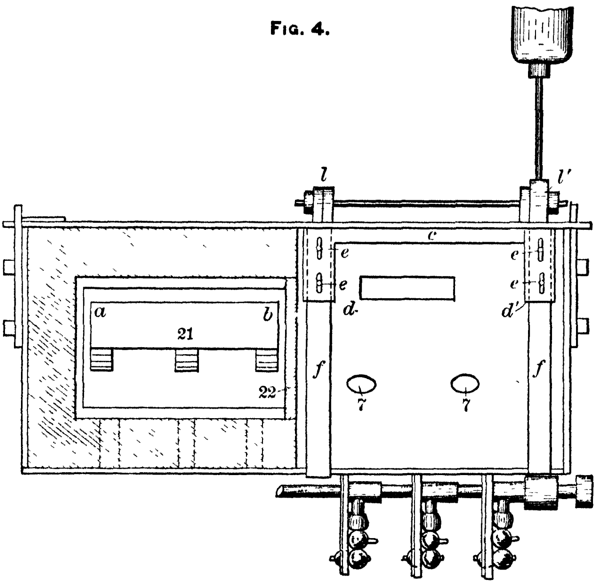


End-Elevation.

the four sides of the furnace together, because, in experimenting with different refractory mixtures, I have hit upon the right composition, which will give a durable, very refractory, non-friable and *non-cracking* tile. The two bands are necessary to hold together the fractured or cracked pieces, which are the inevitable outcome of incorrectly composed and over-baked refractory material. In the top-view, Fig. 4, attention is specially to be drawn to the manner in which the fire-tile forming the lid is held. The tile is 1.5 inches thick, and covers exactly one-half the furnace, including one-half of the diaphragm when the latter is in use. The tile weighs about ten

pounds. It is held by the back-piece *c*, which is cast in one with the eyes *l*, *l'*, and the arms *d*, *d'*. The arms are cast hollow, and have slots *e e*, *e e*. The pieces *f*, *f* are hooked at one end (see *f*, Fig. 3), the hook gripping under the lower edge of the tile in a recess specially molded in the tile. The other end of *f* slides into the hollow arm *d*, having threaded holes which correspond with the slots *e e*. In this way the tile can be quickly inserted and fastened with the set-screws, independently of any slight variations in the size of the tile; avoiding also, as much as possible, heat-radiating iron surfaces. This scheme only works well when the tile is made of the proper

FIG. 4.

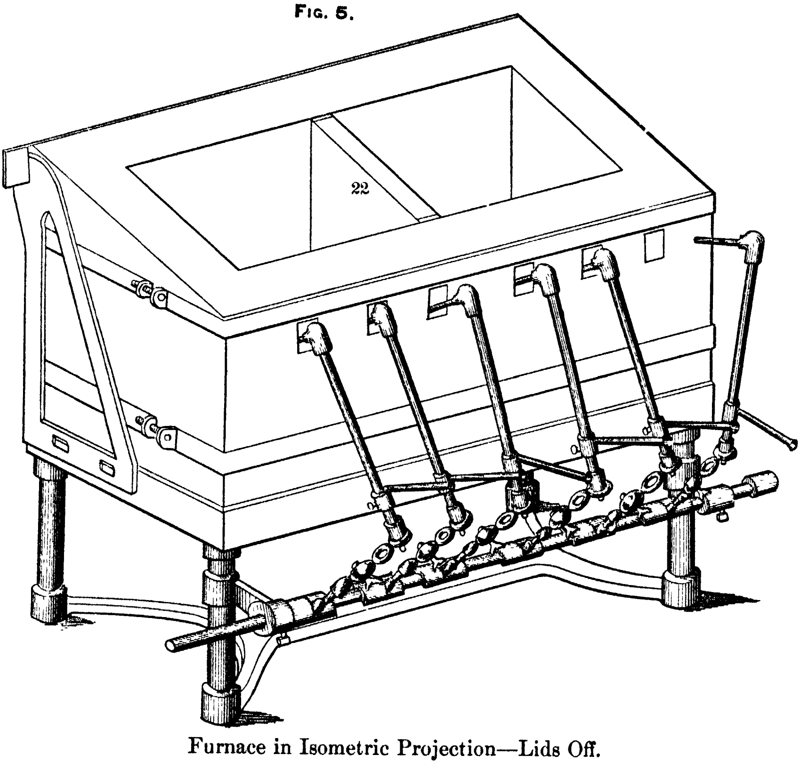


Half Ground-Plan and Half Top-View.

material. Such fire-tiles as are found usually in the lining of portable furnaces will inevitably crack. I have had a furnace in use since last May, and the lids show no sign yet of any crack. The observation holes (7, 7) are so located that the entire interior of one compartment is visible through them. The perforation through the counterweight (10) is made eccentric. After the lid is counterweighted by means of sliding the weight, the nut upon the end of the rod is brought up to the weight. Then, by rotating the weight, the adjustment of the center of gravity is easily performed, a set-screw holding the weight against the rod. Fig. 5 shows the furnace in isometric

projection with lids left off. Fig. 6 shows the scorifier-tongs. A special tool was found necessary, or, rather, convenient, although any goose-neck-tongs of sufficient length can be used. In these special tongs one prong terminates in a circular segment (s), whose radius is equal to that of the scorifier. Above this segmental ring a thin rod (r) is welded to the prong. The length of this rod slightly exceeds the top diameter of the scorifier. The other prong terminates in a flat round piece

FIG. 5.



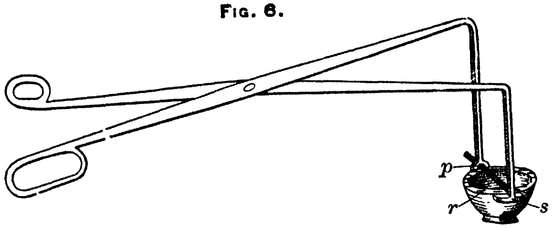
Furnace in Isometric Projection—Lids Off.

perforated by a hole (p) into which the end of the rod r enters, when the grip upon the tongs closes. The rod r coming flat down on the edge of the scorifier prevents a tilting of the latter before the ring and the flat have had time to grip. The operator sits in front of the furnace and looks through the tuyere, behind which the scorifier stands; seeing the tongs, he can adjust the grip. Removing scorifiers from a muffle is simpler, and yet an overtilting is not uncommon. I have not had many mischances, yet I am not quite satisfied, and hope to get, some

time, a more perfect tool. Fig. 7 shows similar tongs, without cross-bar, for the cupels; there is no difficulty in connection with the lifting or setting of the cupels.

Fig. 1 shows at (4) how the Hoskins disk burner is fitted to any furnace. Both burners are set on the same feed-pipe. In working with the gasoline furnaces of Hoskins, every one has the experience that much, if not all, depends upon keeping a

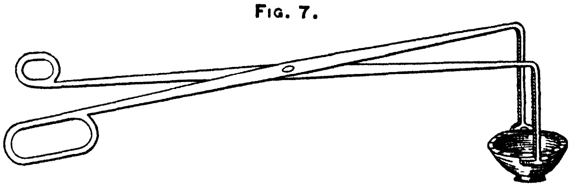
FIG. 6.



Scorifier-Tongs.

strong, as well as a steady, pressure in the supply-tank. The hand-pump is a nuisance. It more than counteracts the relief which the assayer gets from the absence of stoking a coal or coke fire. A hydrostatic pump avoids the whole trouble. I have found the Cleveland Brass Co.'s pump very satisfactory. It is self-acting, and without question the most economical source of pressure, wherever a head of water is at the assayer's disposal.

FIG. 7.



Cupel-Tongs.

The Blast.—Experience has shown that the best effect will be reached when the air-current at the nozzle balances a water-column of 10 inches. The aperture of the nozzle is one-sixteenth of an inch in diameter. The nozzle itself is made from pure nickel wire one-eighth inch in diameter, the block being one-quarter inch long; the sixteenth-inch hole is drilled. This block is driven into the iron nozzle-pipe (8), Fig. 1, while the pipe is hot. The pipe (8) is either straight or curved with a radius of 10 inches. My first nozzles were made by simply

hammering together the end of the iron pipe to one-half its diameter. This answers the purpose for a time. But gradually the oxidation proceeds, reducing the orifice at first, and ultimately making a wider hole, irregular in shape from the peeling or breaking off of the magnetic oxide. The nickel is so much less subject to oxidation that, from present appearances, a very long duration of the nickel-nozzle may be surmised. The air-supply for one or two furnaces is best obtained from a Catalan water-blast when the water-rate is low. I had a Richards pump on hand, and set it up under a head of 120 feet of water. The supply was not sufficient for one nozzle. But since this instrument is primarily intended as a vacuum-pump, and for that purpose is very good, its deficiency as a blast-producer cannot be well laid against it. I replaced the suction-nozzle with a No. 3 boiler-injector, and obtained a supply for three nozzles at 10 inches water-pressure. To do this a very considerable consumption of water is required. For a single furnace or two a pair of bellows will answer, but in this case the assayer needs an assistant to do the treading or pulling. The new assay-laboratory of the Michigan College of Mines, which is just now being equipped, will have sixteen Koenig furnaces. The air will be supplied from a balanced air-holder, into which it will be forced by means of a compressor. The power comes from a 2 H. P. gasoline engine, which will also drive the rock-crusher, rolls, and sifting- and mixing-drums, necessary in preparing the assay-samples. At the assay laboratories of mines and smelters there will be no difficulty, of course, as these works are furnished with air-compressors.

Firing up the Furnace.—The phenomena are just as in a Hoskins furnace. At first there is incomplete combustion and dissociation of the complex carbide molecules. The very disagreeable acetylene is one of the products. But since the furnace is supposed to stand under a hood, the temporary production of that compound, which should not exceed three minutes in duration, is of very little consequence. The acetylene formation is quantitatively stronger when the hot vapors strike a relatively cool surface.

Let the pressure upon the tank be low at the beginning, and if possible keep an extra Bunsen burner, placed right against the rear wall of the furnace under the burner. Such a burner

can be rigged up easily even at a laboratory where gas is not used. An Erlenmeyer flask is filled with 50 cc. of gasoline and loose cotton. The stopper has two perforations. Through one enters a tube to near the bottom, and this tube connects by a rubber tube with one of the blast-nozzles. The other connects with the Bunsen burner. A very strong and steady flame can thus be produced and maintained for a longer time than it will be needed. If such a flame be allowed five minutes to heat the burner-disk, and the gasoline valve be then opened slowly, it will be possible to get the jet to burn at the mouth of the burner at once, instead of burning inside of the tube, and the combustion will be perfect at once, no dissociation-product whatever appearing. It is not necessary to retire the burner-mouth from the furnace at starting. In fact, I have two furnaces mounted back to back, with the four burners in rigid position upon the same pipe-cross; but each burner is supplied with an auxiliary burner. Should the flame strike back into the tube after the furnace is once heated up, the remedy lies in shutting off the supply-valve for an instant, and then opening it very quickly; the auxiliary flame, being partly sucked in at the base of the burner-disk, will then come into play as ready igniter. In fact, I leave this flame, properly reduced, burning as long as the furnace is going, because, as will appear hereafter, it is desirable often to shut down the heat supply completely for a longer or shorter period in order to maintain the most advantageous temperature within. It is just this point which constitutes the superiority of the furnace without muffle over the old muffle-furnace. After roasting one set of assays, for instance, the furnace is at a bright red heat. The fresh samples do not want more than dull cherry-red. Therefore we cut off the gasoline, open the lid, and in five minutes the temperature will be down to the required point. The same thing occurs when, after one batch of crucibles has been taken out, the temperature is a bright yellow. To set the cold crucibles into this heat results generally in cracking one or more crucibles. With this furnace such a risk need not be taken, as the temperature can so quickly be reduced. In cupelling, however, this feature is of paramount importance. I find that from fifteen to eighteen minutes are required to bring the cold furnace to yellow heat.

The Roasting-Operation.—I find cast-iron roasting-dishes preferable to the fire-clay ones. They are square or rectangular, with the corners well rounded to prevent lodging of material. I have them made of one, two and three assay tons' capacity. Their depth is from one-half to five-eighths of an inch. They are furnished with projections by which the tongs can get a good hold without touching the contents. There are covers provided for these dishes to prevent loss from the decrepitation of the pyrite crystals and crystal-fragments during the first period. During this period a slight caking occurs when FeS_2 changes to FeS. I leave the covers on until the blue flame around the further edge of the dish has disappeared. After the lid has been removed I pull out the tuyere-plugs and induce a nearly horizontal air-current by a proper position of the nozzles. The stop-cocks being barely opened, the pressure at the nozzle is much reduced; the current can be made very gentle, and thus a fusion or semi-fusion of high sulphide ores can be easily avoided. Such a regulation is not possible in a muffle, because neither the heat-supply nor the air-supply can be regulated with any degree of nicety. The lid is the real thermostat. Its perfect balance in any position permits a play in depressing or raising the temperature as quick and as unfailing as the stop of an organ or the pedal of a piano. Raking of the ore is done with a hooked wire in the usual way. As the desulphurization progresses, the air-current is increased and the temperature is raised first by lowering the lid, afterwards by turning on more gasoline, until finally the sulphates are destroyed at yellow heat.

The Crucible-Fusion.—Each compartment will hold four Battersea F crucibles. With the diaphragm removed there is room for six, or twelve E, or four J crucibles. The larger number of crucibles will melt down faster than the smaller number of equal size. This is owing to greater absorption in the first place, and increased radiation in the last stage when the high temperature is needed. Working in the usual way, with sodium carbonate as flux, the operator has only to work the lid when the effervescence begins, in order to avoid overflowing of the contents. Watching through the observation-hole, he will see the rising to the rim in one or more crucibles; and as they are all equally hot, or nearly so, the rising in one

will be the rising in all. The opening of the lid fairly flings back the seething masses. Stirring is not needed in any ordinary case. The time is about the same as in a Hoskins furnace, always shorter than in a coke or charcoal wind furnace, but not very much shorter. I have lately experimented in the direction of discarding sodium and potassium carbonates as fluxes in crucible-fusions. The success has been gratifying, and warrants further effort with good promise. The results are not yet ready for publication, because the entire field is not covered. I will say at present only this, that I have put a charge of 1 ton of *quartz* ore, 1 ton of litharge and 3 tons of flux into an F crucible, have put it into the furnace at dull red heat, closed the lid, given full flame, and not looked at it for fifteen minutes. It was then found at quiet fusion. The condition of the crucible showed that the liquid had never risen beyond an inch below the rim. I gave it three more minutes; then poured, and found perfect fusion with easy liquidity, leaving no pellet of lead either in the crucible or in the slag. Every assayer will recognize the technical importance to himself of such a method of fluxing, which gives him an important quarter of an hour to weigh out a fresh lot of samples, or to attend to other necessary work, instead of blistering his face over an ordinary wind-furnace with a stirring-rod, trying to keep the "blasted things" from running over—and often without success even at that, for whilst he stirs one crucible, the others will go over. The future flux will be nearly universal; the difference in the charges will be simply more or less, according to the acid or basic character of the ore. A few exceptional or extreme cases only will remain, in which other fluxing-material will have to be added.

That I should have found the new furnace serviceable for lead-assay work may be expected. The crucibles may either be set upon the bench or on the floor. It might be merely indicated here that I propose to investigate the old Clausthal method by alkaline flux and oxidation; for if its theory be correct, the blow-pipes will give the means for the decomposition of the sulpho-salts. I intend also to try cast-iron crucibles instead of the unwieldy wrought-iron ones; they would be much cheaper than the clay crucibles, but some experimentation will have to precede, in order to establish the proper mixture between white and gray pig.

The Scorification-Assay.—I have reached now the special field of the new furnace and method. The accumulated experience of many generations of assayers leads to the conclusion that, everything else being equal, the assay for silver will come nearest the true amount when the melting down of the charge is brought about in the shortest time. My special experience with the new method brings affirmative evidence towards that doctrine. For in every instance the parallel assay has given a heavier button in the new furnace as compared with the muffle; and since I have found the loss in cupellation to be practically the same in both methods, I am forced to account for the higher result by attributing it to smaller loss in scorification, and here again to the more rapid melting down of the assay. This is, of course, only hypothesis. Perhaps the explanation lies partly in the shorter duration generally. My method in teaching assaying is to give the students half a dozen type-mixtures, mixed up artificially with the greatest care, each sample containing exactly the same quantity of silver (300 ounces per ton), but in a different state of combination, such as chloride, nitrate, sulpharsenate, sulphide, sulphantimonide, mixed with quartz, pyrite, blende, galenite, chalcopyrite, barite, fluorite, calcite, etc. The student puts these samples through until the buttons from each fairly tally. Thus he acquires a real knowledge as to the limits of the method in the special and typical cases. He finds that the loss of silver is not at all the same, but that this loss can vary from 3 to 16 per cent. with the best work possible. The very first assay is made with clean sand, the 30 mgr. of silver, wrapped in lead foil, being placed on top of the charge, sand and test-lead. This gives the minimum loss. During the last spring course the new furnace was first used by the students. There being only one in commission, each man could have it only for one day. A valuable number of data were, however, collected, aside from my own experiments; and upon these data I ventured the previous assertion that the average in all parallel muffle and non-muffle assays was higher in the latter. The first furnace was not mechanically as perfect as the later ones, and I look therefore with special interest towards the coming spring campaign. The fact, however, that the new laboratory will contain sixteen of these furnaces seems to give sufficient warrant that I do not bring this subject before the profession prematurely.

The following is a description of the process:

The furnace is at yellow heat; the bench and the bottom plate have been covered with fine ground brick (less than 40 meshes), which I prefer to bone-ash. The scorifiers have been placed upon the bench, so that each one stands centrally before a tuyere. The bench stands one-half inch back from the tuyere-wall; the rim of the scorifier touches the wall; the tuyeres and the observation-holes are plugged. In three minutes from the closing down of the lid the charge is melted down, forming a steaming bath. Very refractory ores will show, at this stage, floating islands or peninsular formations (the ore-charge being 0.1 A. T.). The tuyere-plugs are removed, and the blast-nozzles are brought into position, so that the jet will strike the circumference of the bath at an angle of about 35°. A very energetic reaction follows, so much so that the fuel-supply has to be cut down about one-half. It will be from 9 to 15 minutes before the bath disappears under the slag. The oxidation, of course, decreases in intensity as the bath becomes smaller, and the flame must be turned up, corresponding to this decrease. The blast-nozzles are now withdrawn, the plugs inserted, and borax-glass is added to the scorifiers, either wrapped in paper or preferably poured from a scorifier used as ladle, held with the tongs. Two minutes later the scorifiers are poured out in the well-known manner. Thus the scorifying-process has occupied altogether, in the mean, seventeen minutes. The most refractory arsenides and antimonides are thoroughly decomposed in this time. I have not observed in any case the formation of matte or speiss, so common in ordinary muffle-work. The A. T. of test-lead has become reduced to a button of from 10 to 12 grammes. With dry ores it is quite unnecessary to keep up the scorification until the bath is covered with the slag. For such ones a duration of twelve minutes is quite sufficient and even desirable. For it is better to remove the lead on the cupel than in the scorifier; the chances of mechanical loss are less.

The Cupellation.—The cupels have been, in the meanwhile, thoroughly ignited, having been disposed on the furnace-bottom between the bench and the back wall, and also piled up in the corners, during scorification. In the matter of cupels I have lately obtained what seems to be a decided improvement,

though, to others, it may not appear in the same light, and this, perhaps, explains why the simple notion has not been adopted in practice long ago. The fact that bone-ash alone will not make a cohering cupel necessitates the employment of some cementing material, in the choice of which there is not much latitude. This suggests at once the idea of holding the bone-ash together in a metallic ring. In the English method of test-cupellation this idea has long been made practical on a large scale. I do not know whether any assayer has tried to apply the iron ring to an ordinary cupel, and hence do not claim any special originality for giving the notion a trial. It may, perhaps, give to others the same satisfaction which it has given to me. The use of sugar or molasses solution I have discarded many years ago, for I consider the inherent viciousness of the practice almost criminal, from a scientific standpoint. We all know that carbon and lead oxide at red heat will either make carbon monoxide or carbon dioxide. We are most careful to ignite our cupels, so that no gases or vapors shall rise from them when once the work-lead is upon them. Yet we put into the very interior of them this multitude of carbon particles, for, indeed, you must ignite a 1.5-inch cupel for many, many hours in an oxidizing atmosphere before the innermost carbon is oxidized. It has been my practice to mix the dry bone-ash with 1.5 per cent. of dry pipe-clay, then to moisten the mixture to the proper degree, and, in moulding, to cover the cupel-mass with pure bone-ash. Whilst this makes a well-cohering cupel, it has the drawback of fissuring if not beaten to the exact point. The present practice is to cut a 1.5-inch gas-pipe into lengths of one inch, to file or grind off the burr, and the containing ring is ready. Moisten the bone-ash very little, press it into the ring with the thumbs, even full, then set on the die, and drive it in even with the rim. One pound of bone-ash makes ten cupels. They can be used at once. They do not fissure, at least I have not yet observed a crack. No danger of breaking of the top rim with the tongs; no mishap has occurred to an assay since I introduced them. At each service a film of black oxide or scale forms on the outside. When the saturated bone-ash is worked out, this film cracks off alone. My experience is not long enough yet to say how many heats a ring will stand, probably between 50 and

100. The cost of the ring does not enter, therefore, as an item of appreciable expense. I find these cupels better absorbents than ordinary cupels, perhaps because the ring acts as a heat-storer and regulator.

Now suppose the cupels have been placed upon the bench, centrally to the tuyere. They should stand off from the wall as much as three-quarters of an inch. They are at yellow heat when the buttons are placed upon them and the tuyere-ports are plugged. A 30-gramme button requires about two minutes to make a smoking bath; smaller buttons less time in proportion. Now the ports are opened, the nozzles are adjusted, and a gentle blast is initiated, until the right point of impact has been found by turning the handle first from left to right, then up or down. The flame is turned off altogether. The correct impact, as well as strength, of the blast will be indicated by the appearance of the bath. The latter looks bright yellow, almost white; it revolves evenly without any oscillations backwards and forwards. The rim of the cupel is dark. This indicates the proper intensity of the reaction. After the operator has gained some practice, a few seconds are consumed only in this adjustment. Passing from one tuyere to the other with the adjustment, the proper time-distance is gained for the glancing. It would not be convenient to have several buttons come to the glance simultaneously. What goes on in the cupel at this stage cannot be expressed by a more appropriate word than "Bessemerizing." The lead proves itself a fuel, and disappears, as silicon and carbon disappear in the Bessemer blow. The lead-fumes pass out of the vent in brown clouds. The lead diminishes its volume, as if it were running through a crack in the cupel. Thirty grammes of lead with ten milligrammes of silver have come to the glance in *four and one-half minutes*, and all this time the heat was shut off from the furnace. This rate of speed, however, is not commendable. Though I have obtained satisfactory results with the high speed, in the general run the silver-results are *too low*. Buttons from gold-assay fusions may be run at this speed without fear of loss. The rate of oxidation is greatest at the beginning, as may be expected, because the air has a larger surface to work on. In the meantime the operator sits on a low stool and watches the progress through the tuyeres. He is not allowed to be idle, however.

He keeps pulling the cupels forward by means of hooked ⅛-inch wire, operating through the tuyere, and at the same time depresses the nozzle. If the buttons are neglected by the operator, the furnace will be at once too cold, and the buttons will wallow in liquid litharge. They do not freeze, as they would invariably do in the muffle under similar conditions. A turning-on of the flame will re-establish the proper gait in a few moments. I find that the best speed for 30-gramme buttons is from seven to eight minutes to the glance. How long a 30-gramme button requires in the muffle every assayer knows, and the large gain in time afforded by the blast-method is evident, not to mention the comfort arising from looking through a 0.75-square-inch aperture into a dull red heat, as against 13 or more square inches of bright redness. To appreciate this, however, one must have felt the difference; words cannot convey the sensation. As soon as the glance is observed, the operator, holding the tongs in his hand, tilts back the lid and lifts out the cupel without rising. The opening of the lid brings with it a lowering of the temperature, and just when the other buttons are nearing the glance this loss is dangerous to their safety. All danger is avoided by turning on the flame at, or, better, somewhat in advance of, the glancing of the first button.

The following may be set down as the essential and distinguishing features of this Bessemerizing of the work-lead:

(1) The cupel is relatively cool.

(2) The lead oxide vaporizes, instead of soaking into the bone-ash.

(3) The speed is from four to six times greater than in the muffle.

(4) The personal comfort of the assayer is greater.

(5) The loss of silver, the so-called cupel-draught, is apparently the same.

In regard to the fifth point, I have proposed to myself a future communication to the Institute, in which I shall attempt an explanation of the evident anomalies in the assay-results, both from muffle-work and from working with my new furnace—anomalies which undoubtedly have occurred to all assayers, without having been accounted for, either to themselves or to the assaying fraternity.

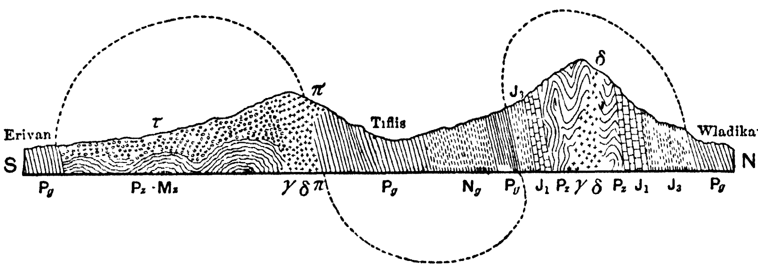
Notes on the Geological Structure of the Caucasus Range Along the Georgia Military Road.

BY DR. PERSIFOR FRAZER, PHILADELPHIA, PA.

(Atlantic City Meeting, February, 1898.)

THE structure of the Caucasus as made out by the Russian geologists and represented in Pamphlet XXII. of the *Livret Guide*, by Loewinson-Lessing, is an overturned anticlinal from Lars to Passanour; a distorted synclinal with monoclinal dip from Passanour to near the summit of the anti-Caucasus chain south of Tiflis; and another overturned anticlinal from this point to Erivan on the Armenian border.

FIG. 1.



SECTION FROM WLADIKAVKAZ TO ERIVAN

(From *Livret Guide*, xxii., p. 3.)

Ng—Neogene; *Pg*—Paleogene; J_3—Upper Jurassic (and Cretaceous in other parts of the chain); J_1—Lias; *Pz*—Paleozoic schists plicated; $\gamma\delta$—massives of granite and diabase; π—porphyries; τ—trachytes, andesites; *Pz*, *Mz*, plicated sedimentary deposits of Armenia.

The details of the geology, however, are extremely complicated—so much so, indeed, as to make it impossible for any set of men to have worked them out in a single season. For the structure as a whole it will be necessary, therefore, to rely implicitly upon the labors of the Russian Geological Survey, and it is intended here only to state these, and to record a few notes supplementing them.

Setting out from Wladikavkaz in the early morning, we pass

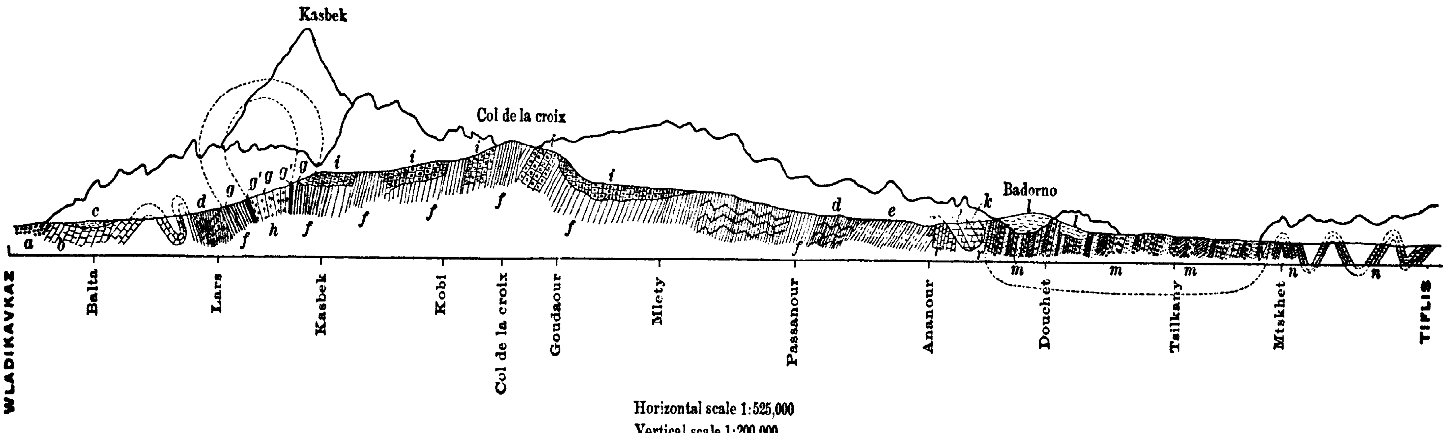
over the upturned edges of the upper Jurassic, which are covered with morainic drift. Once inside the mountain gorge, however, this upper Jurassic is seen on either hand squeezed into irregular folds, of which the axial planes are sometimes inclined to the north and sometimes to the south. At Lars, about 17 wersts (nearly 12 miles) from Wladikavkaz, the highly folded Liassic schists have the deceptive appearance of overlying the Jurassic, owing to an overturn. The Lias extends for a short distance, when the real difficulty of the problem begins. The Liassic schists commence on the north, with moderate dips, and at the point where they are last observed show steeper southerly dips. The Paleozoic schists which succeed them are intercalated first with veins of greenstone, afterwards with veins of gneiss, and again with greenstones; and these show south dips on their northern outcrop, changing rapidly through the vertical to north in a few miles, and implying great compression underground.

Farther south, they show the remains of andesite lava-flows on their upturned edges, and large dikes of andesite between their planes. For about 3 miles south from the first appearance of these Paleozoic schists their structure is fan-shaped, and their planes of lamination dip south on their northern edge, and north from a point some 2 miles south of this to Passanour, or half the distance between Wladikavkaz and Tiflis, say 62 miles.

Beyond Passanour the Liassic schists which were left at Lars with a south dip appear with a north dip, and the upper Jurassic has the false appearance of underlying them, as they falsely seem to underlie the Paleozoic schists which constitute the sedimentary nucleus of the chain. A belt of rocks south of Ananour consisting of Paleogene, and measures not thoroughly determined but suspected to be Jurassic, presents problems so intricate that it is useless to discuss them until the Russian Survey furnishes us much more data. Following these to Tiflis are Miocene and Oligocene deposits, cut at frequent intervals by lavas, to Mtskhet. From here to Tiflis is a series of comparatively regular waves of Oligocene. Between Ananour and Duchet is an extensive deposit of Pliocene conglomerate (*Nagelfluh*).

The following is a free rendition of M. Loewinson-Lessing's chapter on the geology (*Livret Guide*, XXII., pp. 4 and 5).

FIG. 2.



GEOLOGICAL SECTION FROM WLADIKAVKAZ TO TIFLIS

a. Plain of Wladikavkaz; *b*. Upper Jurassic; *c*. Morainic Terrace; *d*. Plicated Lias Schists; *e*. Upper Jurassic; *f*. Paleozoic Schists; *g*. Veins of Greenstone; *h*. Granite; *g'*. Gneiss; *i*. Andesites; *k*. Jurassic (?); *l*. Pliocene Conglomerate (*Nagelfluh*); *r*. Nullipore Limestones and Paleogene Conglomerates; *m*. Miocene; *n*. Oligocene. (From Plate C, *Livret Guide*, xxii.)

Certain argillaceous schists, intercalated with sandstones and quartzite containing some fragments of fossils, have been identified as Paleozoic.

Some calc- and talco-argillaceous schists without fossils have been ascribed to the Lias.

The upper Jurassic contains fossils badly preserved and not certainly identified. On the north slope, its rocks are crystalline limestones, dolomites, and oolitic limestones; on the south slope is a series of compact and variegated siliceous and argillaceous limestones, together with crystalline limestones.

The Paleogene consists of conglomerates and sandstones with the remains of triturated shells, and of Nullipores, together with a series of marls, of gypsiferous sandstone, and lignite between Mtskhet and Tiflis (Oligocene). The Neogene is more varied, and exhibits a series of very diverse rocks of Sarmatic age; variegated marls; limestones, conglomerates, and argillites. One of the beds contains casts of *Helix*.

The glacial deposits are sometimes glacial moraines, sometimes fluvio-glacial deposits with erratic blocks, piled up in terraces on the sides of the Terek valley. As to the alluvions, it is only necessary to instance the pebbles of the beds of the rivers, and here and there muddy loess-like deposits with terrestrial gasteropods.

As to the eruptive rocks, all the recent lavas belong to the andesites, partly amphibolic (Kasbek). The larger part of the pyroxene andesites belong to the enstatite andesites. The andesites are very diverse in their external appearance. They all contain a rhombic pyroxene, and sometimes a little augite. Many types are to be distinguished, based upon the ferro-magnesian constituent.

1. The Kasbek series is characterized by corroded crystals of amphibole and rhombic colorless pyroxene.

2. The series of Sioni belongs to the pyroxene and biotite group.

3. The Goudaour-Mlety series is remarkable for the association of enstatite and macroscopic phenocrysts of amphibole entirely corroded, and pseudomorphs of amphibole in crystals of first consolidation, visible to the naked eye.

Finally the volcanic breccia, the tuffs and the trachytic taxites of Kasbek deserve mention.

The old series of basic vein-rocks is related to the diabases, diorites, metadiorites, holocrystalline green amphibole and diallage porphyrites.

There are also "porphyritoids," especially in the valley of Devdorok, besides Paleozoic beds metamorphosed by the eruptive rock. The greater part of the vein-rocks present a more or less cataclastic and often catalytic appearance. * * *

There remains the granite series. There is a coarse-grained amphiboliferous granite, some varieties recalling protogine; taxitic rocks (*schlieriger Granit*); finally gneiss and greisen. Some specimens of granite show cataclastic phenomena more or less strongly. The gneisses are all very cataclastic, and are in great part "metagneisses" or particularly "clastogneisses."

The New Breaker at Cranberry Coal-Mine, Hazleton, Pa.

BY W. S. AYRES, HAZLETON, PA.

(Atlantic City Meeting, February, 1898.)

The construction of a new breaker at the Cranberry Colliery, Hazleton, Pa., was forced upon the operators, A. Pardee & Co., by a fire which destroyed the entire plant early in January, 1896.

The loss included, in addition to the breaker, three boiler-houses and three engine-houses.

The boilers were not injured, but the steam-pipes and connections were all broken and warped by the fire. The boilers were probably saved from injury by the presence of mind of the fireman, who started all his feed-pumps and filled the boilers with water.

The hoisting-engines were more or less warped and cracked, but by patching and replacing the injured and broken parts, they were made to do satisfactory service in the new plant. It may be worthy of mention that the wrought-iron shaft of one of the hoisting-drums, 12 inches in diameter by 17 feet long, was so heated in the fire that the weight of the drum caused it

to sag 3½ inches at the center. This shaft was taken out, placed on iron skids, and heated red-hot by building a fire around it. Then by the use of a drop, made of a 12- by 12-inch by 10-foot stick of timber, it was straightened to a surprising degree of accuracy. It was put back into the drum, and is running satisfactorily without any further expense being put upon it.

The breaker-engine and all the machinery in the breaker, including the rolls, screens, jigs, etc., were a total wreck. Many curious freaks were seen in clearing away the ruins. A mention of two will suffice. A shaft 6 inches in diameter was bent over its pillow-block to a right angle, with a sharp corner in the angle. And again, the main pulley on the counter-shaft, which was about 36 inches in diameter, with a 24-inch face, had its hub and spokes melted out, and where the rim tilted against the shaft it was also melted and united to the shaft, as one piece, by a perfect weld between cast- and wrought-iron, which could not be separated.

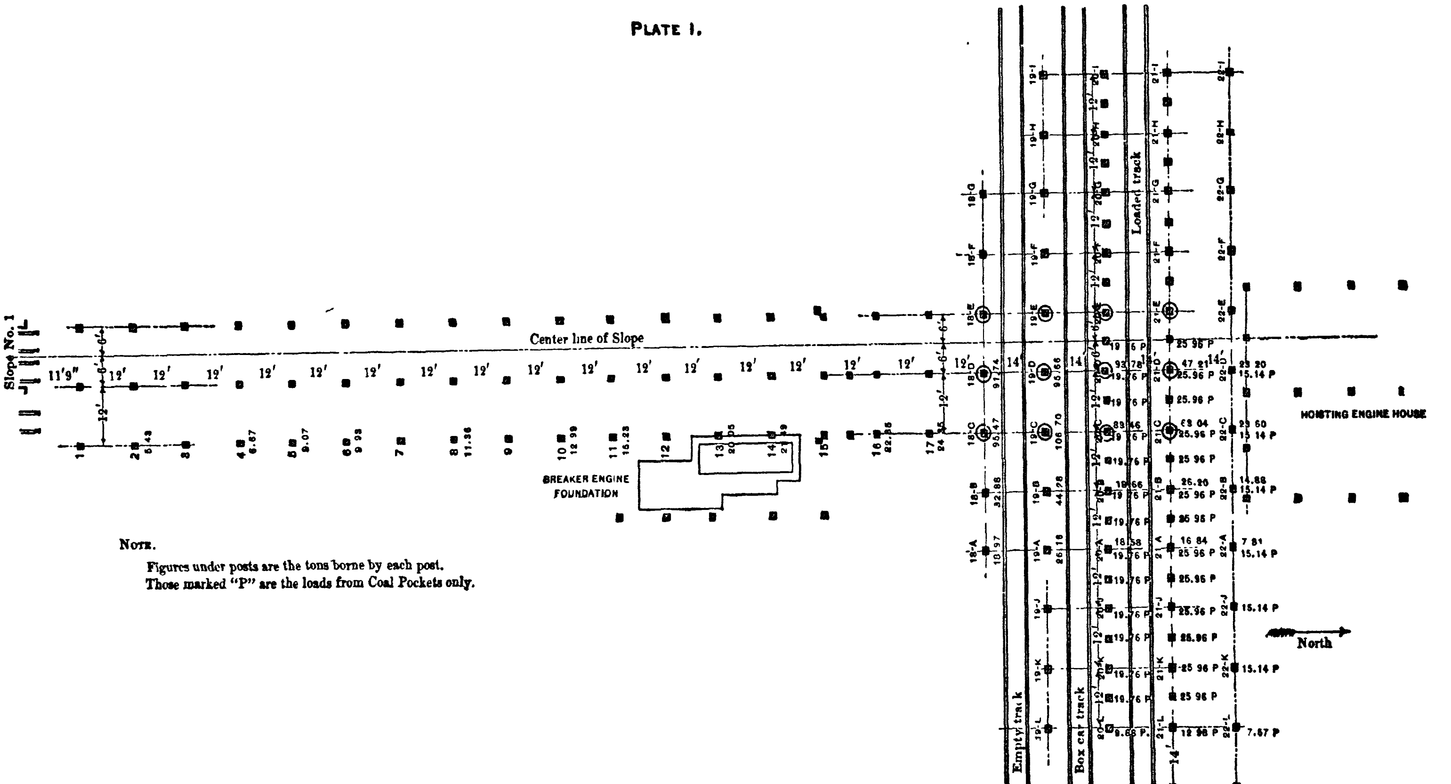
This wholesale destruction made it necessary to begin at the beginning to rebuild. In many ways a complete destruction often proves to be a benefit to the operator. Usually, machinery that has been only slightly damaged, yet did not have the capacity desired, or did not do its work as well as later designs, is restored and put into the new plant, on the plea of economy, with the result that the new plant is an old one to start with. At Cranberry, so far as the coal-preparing machinery was concerned, we were relieved, by a total destruction of the old machinery, from any such considerations.

There were some limitations, however, and within them many desirable results were to be attained.

Among the things aimed at in the design of the new breaker were:

1. An arrangement and style of machinery that would be very simple, and enable the coal, as it comes from the mine and is hoisted to the top of the breaker, to descend by gravity through the various rolls, screens and jigs to the storage-pockets without the use of any elevator or device for rehandling. Also, such designs of crushing-rolls as would produce a larger percentage of the larger sizes of coal, and a correspondingly smaller percentage of the smaller sizes, than had been

PLATE I.



NOTE.

Figures under posts are the tons borne by each post.
Those marked "P" are the loads from Coal Pockets only.

FOUNDATION PLAN, CRANBERRY BREAKER.

obtained in the old breaker. The special arrangement adopted is the conception of Mr. F. Pardee, the Superintendent.

2. The introduction of easy grades and turns in all the chutes that convey the coal from one machine to another and to the pockets, so as to avoid chipping and waste.

3. A design of frame for the breaker that would be very rigid, and at the same time use the smallest amount of material for the strength and rigidity required, and that would also embody the possibility of taking out any piece and replacing it, as, for instance, in repairs.

The detailed description of the breaker now to be given will follow, not the above order, which is the logical order controlling the design, but the successive steps of construction, namely: (1) foundations; (2) frame of structure; (3) machinery; (4) chutes and connections; to which will be added (5) a statement of results.

Foundations.

The foundations were built of stone laid in cement, one foundation for each post, having a broad base and tapering to a capstone about 2 feet square. There are, in all, 131 of these pyramidal foundations. The size of the base was determined in each case by the nature of the soil and the load that would come upon the post. A careful calculation was made of the load that would come upon each, comprising the weight of the structure, machinery, coal going through and stored in it, the water in jigs, and the effect of vibrations, wind and snow. The foundation-plan, Plate I., shows the results of this calculation in tons of 2000 pounds. The figures under the posts give the number of tons borne by each. Those marked "P" are the loads from the coal-pockets only.

Fortunately, the majority of the foundations under the breaker proper, including all of those carrying the greater weights and subject to the greater vibrations, were upon rock. Where they came upon clay and hard-pan,* one to two tons pressure per square foot was allowed, and the size of the base for each was figured from the weight to be carried by that post.

These foundations were not less than four feet deep, so as to

* See Rankine's *Civ. Eng.*, Art. 238, page 380.

ensure them against frost. Yet at this shallow depth, the pressure per square foot assumed for each was far less than the greatest intensity consistent with the stability of the earth. To allow fully for the vibrations that would be caused by the machinery, when in motion, an intensity of pressure was assumed, representing about half that allowable for the same depth were the structure to remain in absolute repose.*

The cement used was the Hudson River Rosendale, mixed, one of cement to two of sand. The stone used was a very hard conglomerate, having a probable safe working resistance to crushing of at least 50 tons per square foot, this being one-tenth of its probable ultimate strength. To avoid any settling, a bed of about two inches of sand was first put into each pit, and the first course of stone was well rammed into it. Large flat stones were selected for this course. In each successive course great care was taken that each stone should be well bedded upon the one below. Only the hammer was used, however, in dressing, so that stone touched stone with only enough cement between to fill up the irregularities of the bedding-faces and abutting joints. This precaution was necessary, as the cement has a safe working strength of only 3 to 5 tons per square foot, while the stone, even though it touches the one under it on only a few small surfaces, has a capacity to resist 500 tons per square foot before crushing. That is to say, in all foundations carrying as much as 125 tons, the small surfaces of contact together must equal an area not less than $\frac{125}{500} = \frac{1}{4}$ of a square foot, or 6 inches square.

This area is quite easily secured in ordinary hammer-dressing, and, when reinforced by the resistance of the cement, the foundation can be relied upon.

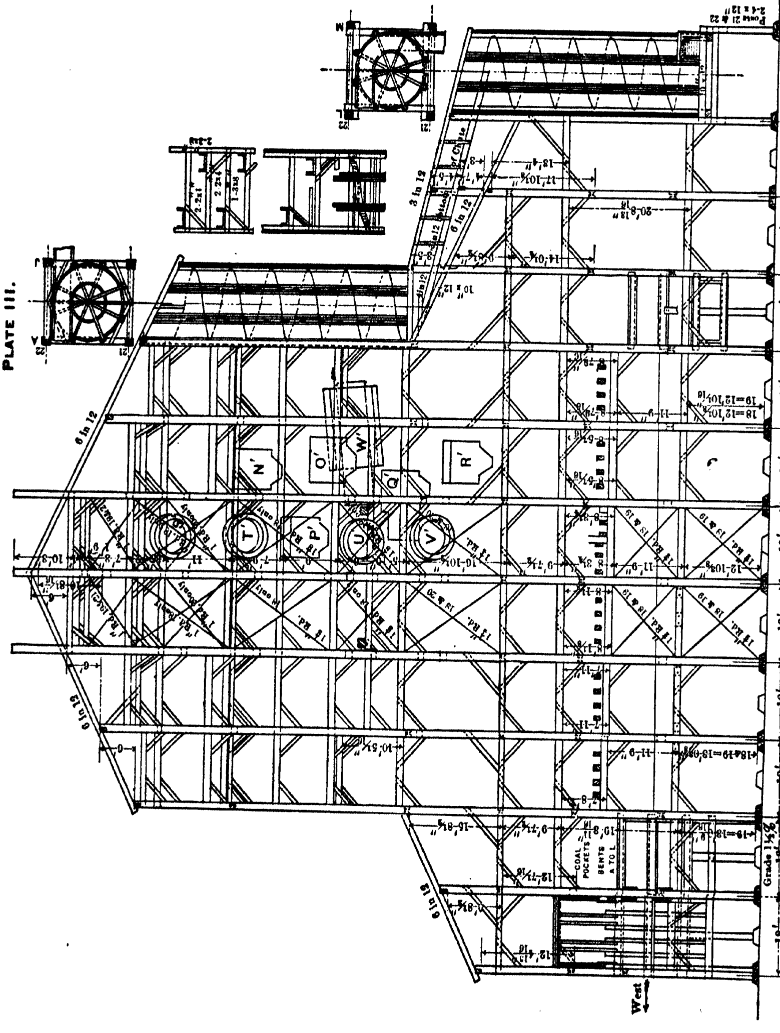
The cap-stones were from 20 to 24 inches square by 12 to 15 inches thick, rough-chiseled to a level surface on top, after bedding. These caps, on foundations 18, 19, 20 and 21, C, D and E, were chipped to a fixed level, to receive 12 steel columns (to be described under the head of "Frame of Structure"), while all others were chipped to the nearest approximate level consistent with a true surface, and the wooden posts were cut longer or shorter, to accommodate them to these deviations from the fixed levels.

* See Rankine's *Applied Mechanics*, Art. 198, p. 219.

PLATE III.

EXPLANATION

N' 4-BROKEN COAL JIGS
O' 4-EGG " "
P' 4-STOVE " "
Q' 4-CHEST. " "
R' 2-PEA " "
S' 4-STEAMBOAT AND BROKEN SCREENS
T' 4-EGG AND STOVE SCREENS
U' 4-CHEST. AND PEA SCREENS
V' 4-BUCK. AND NO. 2 BUCK. SCREENS
W' 2-EGG, STOVE, CHEST. AND PEA SCREENS



CROSS-SECTION THROUGH BENT NO 18, CRANBERRY BREAKER, LOOKING NORTH.

The above method was adopted in the construction of the foundations, as giving the least risk of settling, the least mason-work, the least cement, and the least labor.

FRAME OF STRUCTURE.

The old breaker, which was consumed by fire, had been constructed entirely of timber, and according to the old plan for wooden structures, which has been in general use in the anthracite region since the mines were first opened, fifty years ago. It consisted of solid members, braced with the usual short braces, framed together in the usual way with mortises and tenons, and pinned with 1¼-inch wooden pins. All upright pieces were set upon, and mortised into sills, which rested upon long stone-wall foundations. The old structure contained about 1,300,000 feet, B. M., of lumber.

The defects developed in the old structure were:

1. The foundations would settle directly under the posts for want of a base of sufficient size at these points, while between posts the foundations were unaffected and useless.

2. The sill-timbers would crush and rot, and the posts would also rot, causing a very great settling, which, added to the settling of the foundations, would throw the machinery all out of line. This was a very serious matter.

3. The mortises and tenons, particularly those of the braces, formed pockets for water to lodge in. When the pin or tenon was rotted off, only the boxing prevented the timber from falling out.

4. It was impossible, except at relatively great expense, to replace any timber or brace framed with mortise and tenon.

5. In the solid members, such as posts, ties, inter-ties, etc., the center or heart of the stick was found in many cases to be entirely rotten, even to within an inch or less of the sides.

6. There was a want of rigidity in the structure, due to the short corner-braces, and to the tendency of those pieces acting as struts to shove apart the joints between ties and posts.

7. The strength of any post depended entirely upon what was left of its cross-section after cutting the mortise.

To meet the evils above enumerated, to avoid the use of large and long timbers, which are becoming more scarce and expensive every year, and to use the timber in its most efficient

directions, was the problem; and it was solved in the following manner:

The first of the above evils was met by building an individual foundation for each post, with a base sufficiently large to resist the pressure without settling, as described above under the head of "Foundations."

The second was avoided by not using sills under any part of the structure; and the considerations which suggested remedies for this and for the third, fourth, fifth, sixth, and seventh defects, were based upon well-known facts concerning the strength of timber, namely:

a. The ultimate resistance to crushing of white pine, for example, across the grain, is only about 550 pounds per square inch.

b. The ultimate resistance to crushing in the direction of the grain, or end-wise, is about 5500 pounds per square inch.

c. The ultimate tensile strength in the direction of the grain, or end-wise, is about 11,000 pounds per square inch.

d. A hollow column is stronger than a solid one of the same cross-sectional area of material; the strength increasing as the hollow space increases.

From these four facts it is plain:

e. That a stick of white pine can resist in the direction of its grain, or end-wise, *ten times* the crushing force that it can resist across the grain.

f. That a stick of white pine can resist in the direction of its grain or end-wise, *twice* as much tensile as crushing-force.

g. That a post made hollow in some way would require less timber to carry a given weight than would be required if it were made of one solid piece.

Other kinds of timber, such as oak, yellow pine, hemlock, etc., have ultimate strengths differing from those of white pine, but they follow the same general law, and the differences affect only the ratios given under *e* and *f*.

In view of these facts and deductions, it was decided:

1. Not to place any timber in such a position that it would be acted upon by the load across the grain either in compression or tension, except when this was absolutely necessary, as in the case of bearers under tracks and machinery.

2. To make all braces, as far as possible, act in tension.

3. To build up the posts from ordinary commercial sizes and lengths of lumber, such as can be found in almost any lumber-yard, and to make them hollow.

These points decided, the next step was to design and arrange the machinery in such relative positions, without any reference to the frame, as would accomplish in the best manner the preparation and handling of the coal. Two thousand tons per day of 10 hours was the capacity aimed at.

This done, an outline-design of the structure, having a total height of 200 feet, was sketched about the machinery. Then a very careful calculation, which covered about 100 pages, was made to determine the strain in each piece of the entire structure. This calculation embraced the weight of each machine, the coal running through it, the water in the jigs, all connections, platforms and enclosures, the weight of the roof and snow, the strains due to the wind, and the weight of each piece of timber that entered into the structure. The calculations of the wind-strains in this high structure were complicated, and in some members showed a possible strain of 27.5 tons to be provided against, assuming a wind-velocity of 80 miles per hour.

After the general outlines of the frame and its bracing were sketched around the machinery so as to accommodate every convenience of operation, strain-sheets were constructed from the calculations, and the timbers were assigned to resist the various forces, according to the well-known principles above stated.

The result of this by no means trifling labor is shown in the elevation and section of the breaker, Plates II. and III.

The size of each post and strut was determined by Gordon's formula, using a factor of safety of 10, and that of each tie and brace by using one-tenth of the ultimate tensile strength as the safe working-load. To the net sectional area so determined was added whatever would be cut away by the bolt-holes, bolts being used throughout to connect all joints. The number and size of bolts required for each joint was determined by dividing the total strain on the member in consideration by the load one bolt would carry, this load being found by multiplying together the thickness of the timber, the diameter of the bolt and the safe working-pressure per square inch for the timber used.

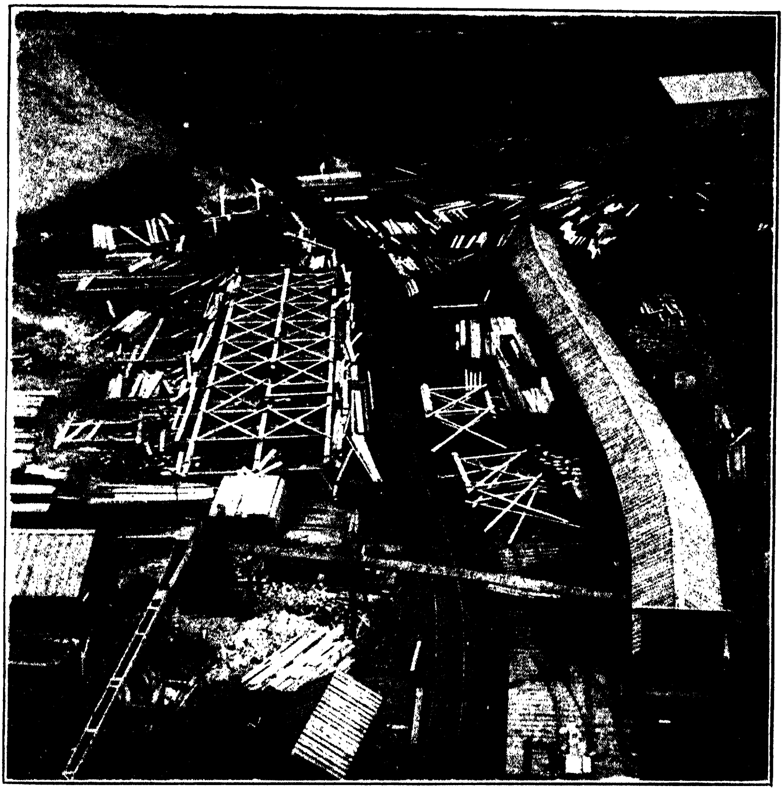
Those timbers which would receive severe shocks (as in the dump, under the rolls, etc.) were increased in size to at least twice the strength required for a quiescent load.

In selecting timbers for each place, only stock or commercial sizes were used. The obvious reasons for this were that stock sizes are cheaper and are always obtainable, even in emergency; also, that they can be more closely inspected, and that they will season better and dry out quicker after being wet.

From the elevation shown in Plate II., it will be observed that the structure is in two principal divisions; bents 1 to 17 inclusive forming the plane, while bents 18 to 22 inclusive form the breaker proper.

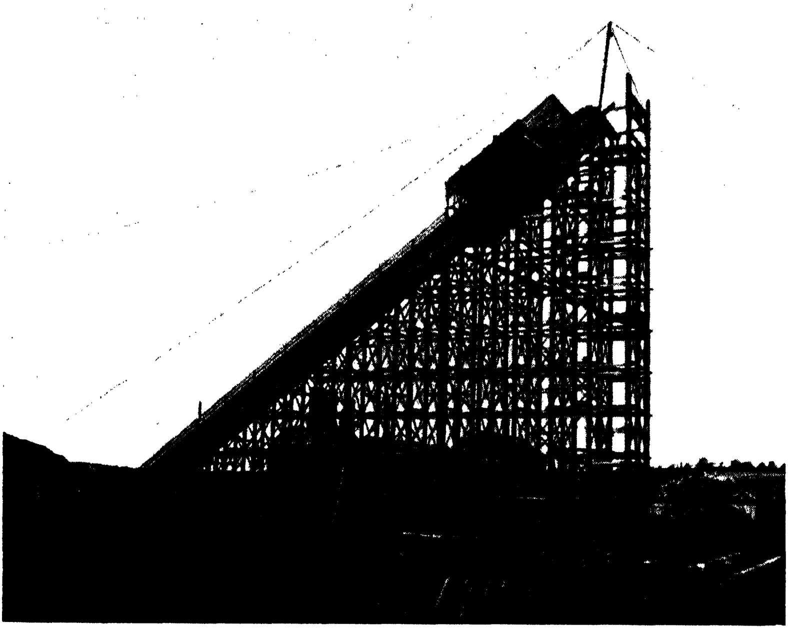
The plane has three tracks with room for the fourth. Each bent is framed as shown in the section through bent No. 8, shown on the left of Plate II. The questions of rigidity and of stability against wind-pressure were the most important in designing the plane. To make the structure with sufficient width of base to resist the wind-pressure would require a base so wide that it would occupy valuable ground desired for other purposes. At the same time, the tops of the high bents would have an unavoidable sway, which increases, as shown by experience, the longer the structure is in use. The simple narrow-base parallel bent, shown in Plate II., with guy-lines to meet both the sway and the wind-strains, was decided upon. The higher bents were provided with two sets of guy-lines, one set attached about 70 feet from the ground, and the other at the top. The wind-strains on these guy-lines, with the wind at 80 miles per hour, is estimated to be 10 tons on each. These guy-lines were grouped in twos and fours at the anchors, which consisted of old railroad-rails, bent like the letter A, with a hook turned on the lower end of each leg, into which was laid a piece of old shafting, and, across the shafting, pieces of old mine-rails. On top of the rails flat stones and pieces of slate from the mines were laid to the depth of 1 foot; and then the earth from the pit was thrown in. Each pit was made large enough so that the weight of earth on top of the anchor would exceed the combined pulls of the guy-lines attached to that anchor. A screw-device for adjustment was attached to each guy-line, and formed the connection with the anchor. These guy-lines are of $1\frac{3}{8}$-inch and $1\frac{1}{4}$-inch galvanized steel, and were

PLATE IV.



Bent No. 13 on the Skids, Cranberry Breaker.

PLATE IV. A.

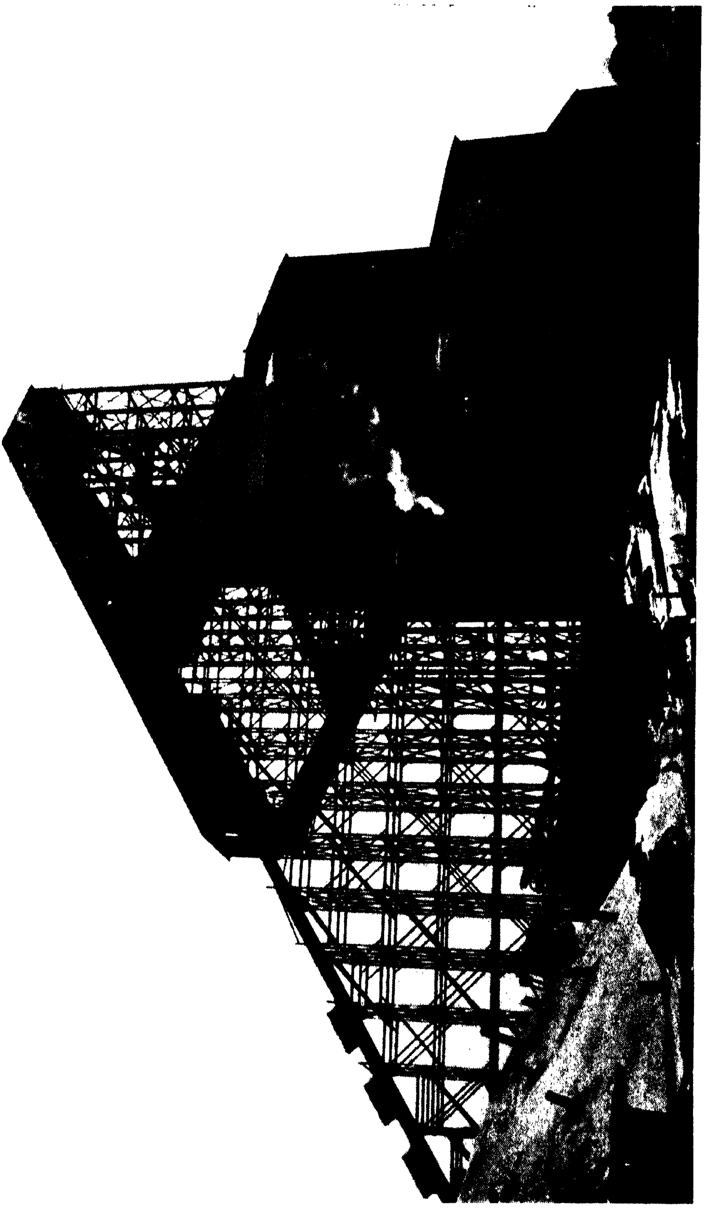


Bent No. 17 in Process of Erection, Cranberry Breaker.

PLATE IV. B.

Steel Column in Process of Erection, Cranberry Breaker, Looking East.

PLATE IV. C.



View of Cranberry Breaker Completed, Looking West.

PLATE IV. D.

View of Cranberry Breaker Completed, Looking Southeast.

finally adjusted so that the deflection taken out of them by a wind having a velocity of 80 miles per hour would not allow the structure, when struck by such a hurricane, to sway more than one-fourth of an inch at the height of 165 feet. A hurricane having a 60-mile gait did strike it, July 27, 1896, but without producing any appreciable sway.

The mode of connecting both the lower and upper guy-lines to the posts is shown in detail on Plates VI. and VII.

The posts of the plane are made of two pieces separated 6 inches, and are butt-jointed every 14 feet. These two pieces are 3 by 10 inches for bents 1 to 9 inclusive; 4 by 10 inches for bents 10 and 11; and 4 by 12 inches for bents 12 to 17 inclusive.

The cross-diagonal braces are 3 by 6 inches in size, and are entered between the post-pieces, as shown in the elevation of bent No. 8 on Plate II. The longitudinal diagonal braces are 2 by 6 inches in size. Owing to the slender proportions of these braces they will practically act only in tension,—a condition which, coupled with the arrangement of the braces at the joints, will pull the joints together, and never tend to separate them. This completely removes the objectionable features of the short brace, and at the same time makes each panel a rigid frame, which the short brace cannot accomplish.

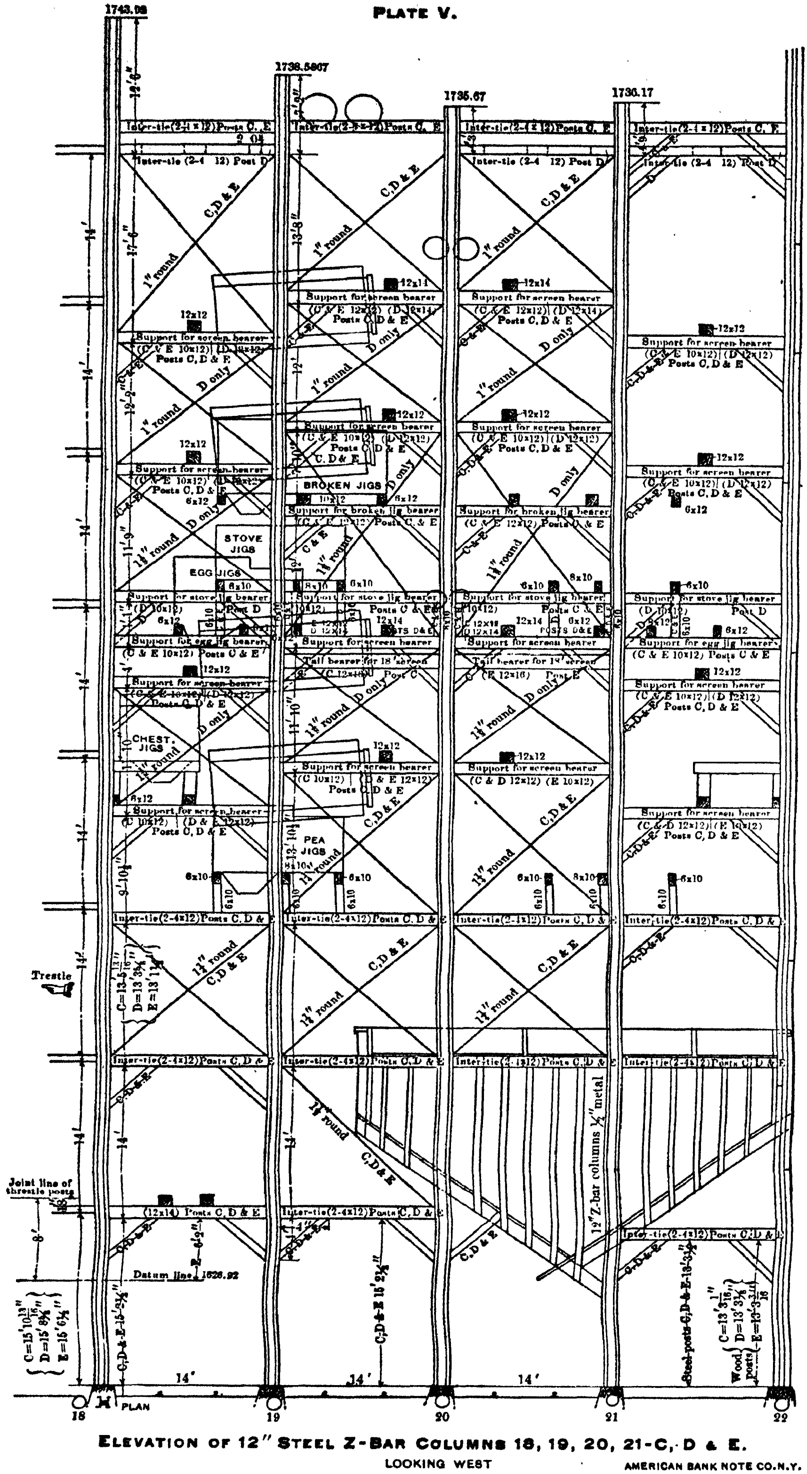
The cross-ties and inter-ties are made of two 2- by 8-inch pieces, separated by fillers and joint-pieces. In the cross-ties they are separated 6 inches throughout, while in the inter-ties they are separated 4 inches at the center and only 2 inches at each end.

The mode of framing employed by the contractor was to lay each bent upon skids, placing the various posts, cross-ties, and cross-diagonal braces in their exact positions, as shown in the view of "Bent No. 13 on the Skids," Plate IV.

The work of framing consisted in simply sawing the post and cross-tie pieces to a given length, sizing the post-pieces, where the cross-ties came against them, to a uniform distance from the center line of the post, and then laying the whole on the skids, as above described, and boring through for the bolts.

All the boring was done by hand, but certainly many hundred dollars would have been saved to the contractor if he had used, as was suggested to him by the writer, some portable

PLATE V.



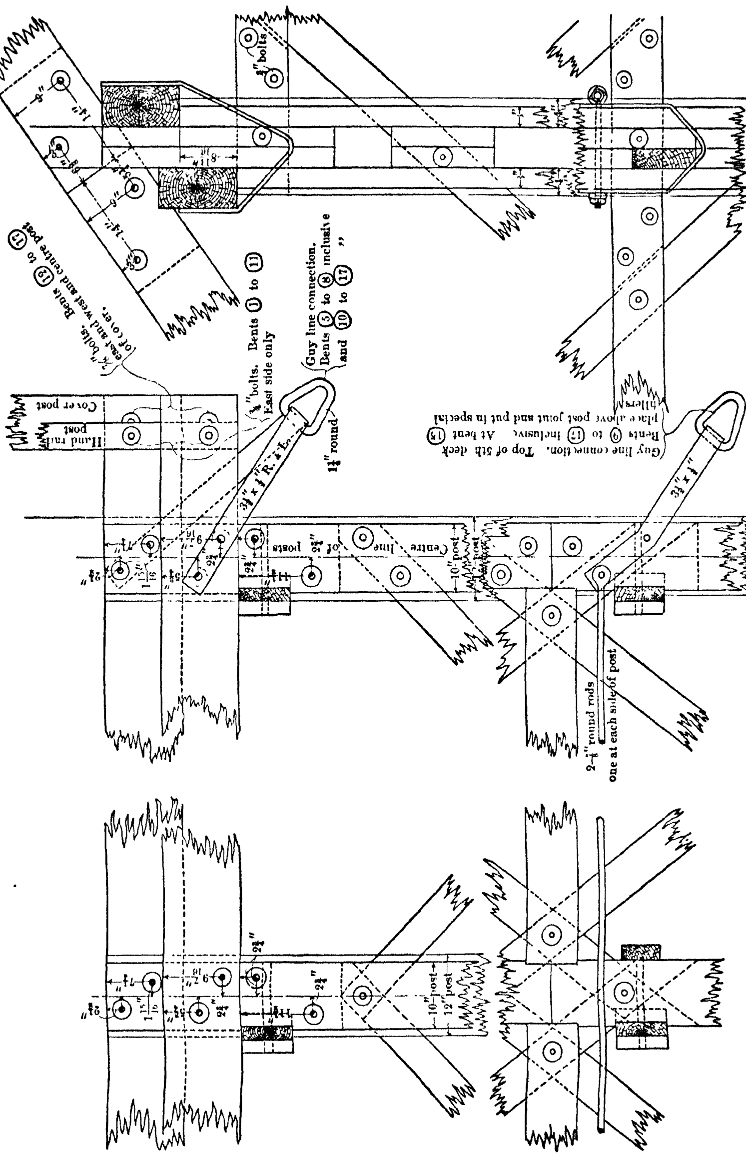
ELEVATION OF 12" STEEL Z-BAR COLUMNS 18, 19, 20, 21-C, D & E.

LOOKING WEST

AMERICAN BANK NOTE CO. N.Y.

power-boring-mill, such as that made by the Stow Flexible Shaft Co., of Philadelphia. This boring-machine, complete, costs about $125.

PLATE VI.



DETAIL OF GUY LINE CONNECTIONS, CRANBERRY BREAKER.

After boring the holes, all bolts were put in, except those, the omission of which would permit the bent to be taken apart in sections for erection. Four of these sections are shown in the view of bent No 13, Plate IV.

The details of the joints are shown on Plates VIII. and IX.

The method of erecting is shown in Plate IV., A.

The plane just described practically continues beyond bent 17, and on top of the breaker proper, from bents 18 to 21 inclusive. The highest part, between bents 20 and 21, carries the three main sheaves for the hoisting-ropes.

The strain coming upon these three sheaves, amounting to more than 30 tons, is carried directly to the foundations by two struts fastened to the outside of the posts, as shown in Plate II. In like manner the main counter-shaft is provided with a vertical strut to resist the jar of the bevel-gears, and with one running diagonally to the engine-foundation, to resist the weight of the shaft, its equipment, and the pull on the driving-ropes, together amounting to 8 tons; also with three horizontal struts, forming the bearers for the counter-shaft, and abutting against the three rows of steel columns in the breaker, to resist the pull on the driving-ropes running from the counter-shaft to the rolls. The object of these devices was to carry the ever-varying and intense strains produced at these working-points directly to the foundations; and the wisdom of it has been shown by the fact that the structure is entirely free from any racking strains or vibrations.

From the above description of the plane, and from an inspection of Plates II., IV. and VI. to IX., showing how the parts are joined in the structure, it will appear that all pieces are of easy access for repairs. For example, to replace any section of a post, the entire weight coming on that post can be taken off by simply slacking the guy-line on that side and tightening the one on the opposite side; and then, by removing a few bolts, the piece is easily removed.

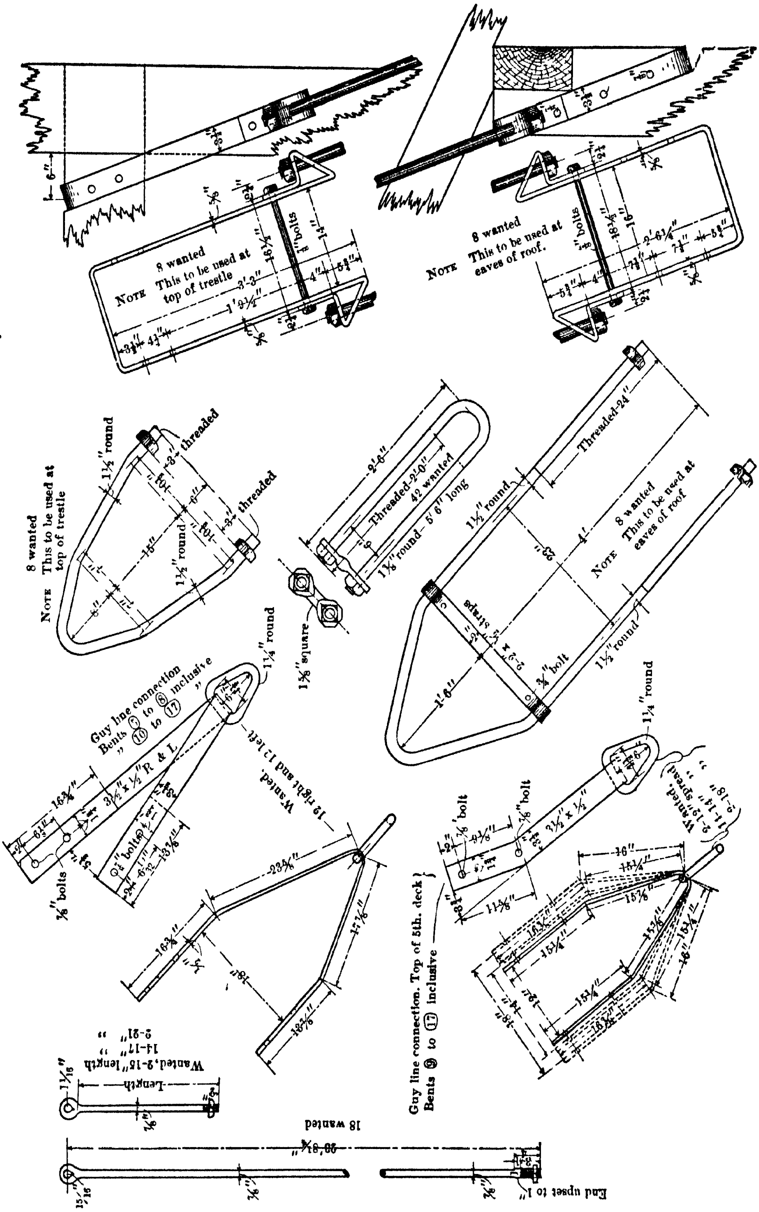
The breaker proper was framed in the same general style as the plane, although some modifications of the cross-ties, inter-ties, and bracing were found necessary about the machinery.

The majority of the posts are made of two 6- by 12-inch pieces of yellow-pine, separated 4 inches.

The cross-ties and inter-ties are made of two 4- by 12-inch pieces, separated 4 inches. The fish-pieces, in the cross-ties which come directly opposite each other, run between the post-pieces, and, as one piece, hold the ends of the opposite cross-ties against the post. In cross-ties not opposite, the fish-piece stops at

the opposite edge of the post. The inter-ties, all of which come against the side of the 6- by 12-inch post-pieces, are cut square

PLATE VII.



DETAILS OF GUY-LINE FORGINGS, CRANBERRY BREAKER.

and between the posts, and are carried by a cast bracket which forms the connection with the post. This bracket is shown on Plate X.

On posts 18, 19, 20 and 21, C., D. and E. (see plan of foundation, Plate I.), the load to be carried is so great that a compact post 2 feet square would be required for each. A compact or solid post 2 feet square was out of the question. Its most simple form would be four 12- by 12-inch sticks close together,—a form that would be sure to rot. Besides, a post of this size would take up too much valuable space. Therefore, for these twelve posts, steel columns of the Carnegie 12-inch Z-bar type were substituted. These twelve columns carry the greater part of the machinery, and are braced with diagonal braces. They extend from the foundations to the crusher-rolls, and vary in length from 118 to 126 feet. They are shown in elevation in Plate V. and also in the process of erection in Plate IV., B. While they cost erected about one and seven-eighths times the cost of the equivalent wooden posts, yet they have the advantage of greater rigidity and greater durability, in addition to occupying much less valuable space.

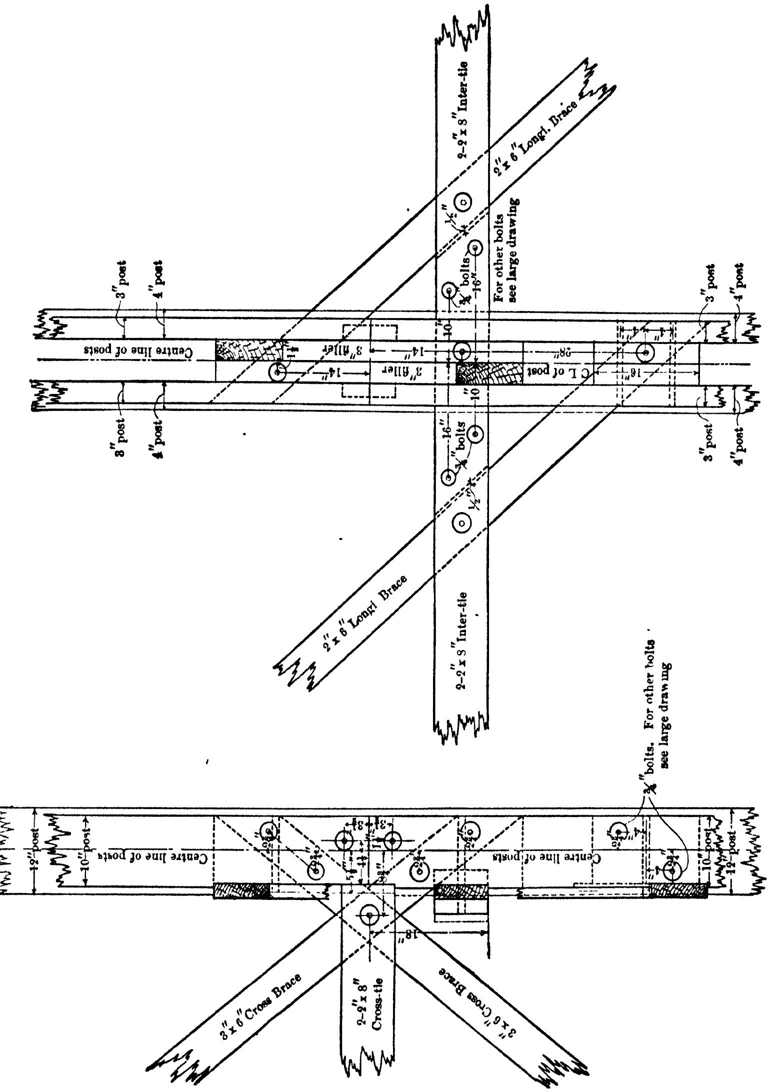
The framing of the coal-pockets, eleven in number, for receiving the seven different sizes of prepared coal, each having a capacity of 100 tons, is such that they rest on posts entirely independent of the posts of the breaker. This was done chiefly to avoid adding these 1100 tons to the already great load of the breaker, and secondly, because it was far easier to provide for their support by using independent posts, giving at the same time far greater advantages for repairs. The details of this framing are shown in Plates II., III. and XII.

Other details of special parts in the breaker will be briefly described in connection with the description of the machinery. Before passing to that subject, however, it may possibly be of interest to call attention here to the construction of the roof-trusses of the cylinder-boiler house and of the hoisting-engine house, shown in Plates XIII. and XIV. They are made entirely of plank fastened together with wire spikes. The number and lengths of spikes required for each joint are written on the designs. Two carpenters and four laborers put these trusses in place, one plank at a time. Each truss of the boiler-house will sustain 3 tons at each of its two principal joints on the lower chord, and that of the engine-house, 6 tons. The boiler-house is 53 by 134 feet in size, covering 25 cylinder-boilers.

The old boiler-house was of the solid-timber type, and con-

tained 48,000 feet, B. M., of lumber, while the new structure contains about 23,000 feet, thus saving 25,000 feet over the solid-timber type.

PLATE VIII.



DETAIL OF OUTSIDE POSTS OF SLOPE PLANE, CRANBERRY BREAKER.

Machinery.

Concerning the machinery, the things aimed at, as already stated in the first part of this paper, were "an arrangement and style of machinery that would be very simple, and enable the coal, as it comes from the mine and is hoisted to the top of

the breaker, to descend by gravity through the various rolls, screens and jigs to the storage-pockets without the use of any elevator or device for re-handling. Also, such designs of crushing-rolls as would produce a larger percentage of the larger sizes of coal, and a correspondingly smaller percentage of the smaller sizes, than had been obtained in the old breaker."

The relative arrangement of the dump, rolls, screens, jigs and pockets, adopted for the accomplishment of these aims, is as follows:

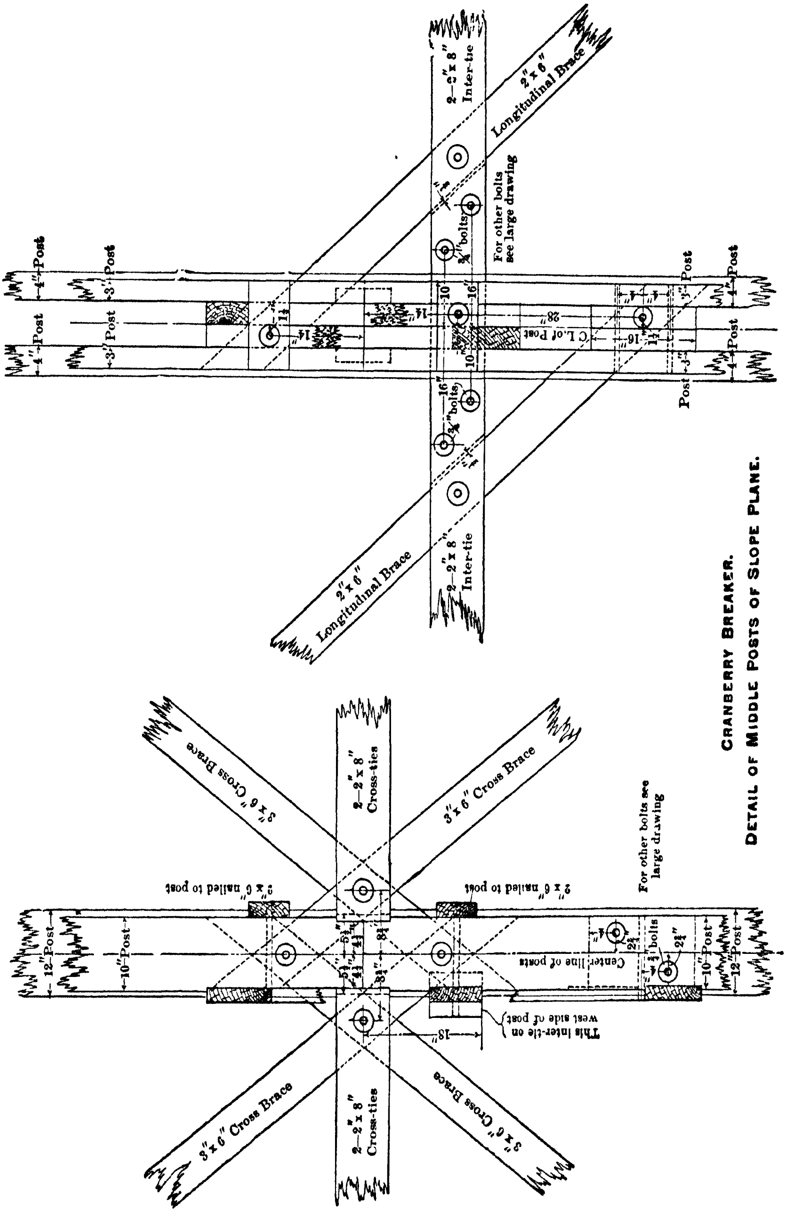
All coal is hoisted to the top of the breaker direct from the bottom of the mine in 2-ton mine-cars, and is there dumped into a dump holding 25 cars. This storage-capacity is to insure a regular supply of coal for the breaker against any irregularity in the hoisting. From this dump the coal passes over bars, to sift out the fine coal, and the larger pieces are fed into the crusher-rolls by a regulating feed, or, if lump-coal is to be made, another set of bars is put in over the rolls, and the suitable lumps of clear coal are pulled aside and passed to the lump-coal chute, all other pieces being passed into the crusher-rolls. Just below these rolls the crushed coal and that coming from the bars are mixed, and the mixed product is divided into two equal streams, one going to the prepared-rolls of the east side of the breaker and one to those of the west side; or the whole product can be turned to either set, as desired.

From this point down the breaker is divided into two distinct parts, exactly alike, each a complete breaker within itself. This is to enable work to continue in one-half of the breaker if a break-down should occur in the other half, or, in running on half-time, to run half of the breaker all of the time.

The product from each set of prepared-rolls is again divided immediately under the rolls, half going to a complete set of screens on the north side and half to a complete set on the south side. At this point, therefore, when both sides are running, the total product is divided into four equal parts, two parts from each set of prepared-rolls. Each of these four parts passes directly into a screen, the "over"-product of which is steamboat and broken sizes. The steamboat-size passes to a picking-space and then to the egg-rolls. The broken size passes to a jig, then to a picking-space, and then to either the pocket for that size, or, if not wanted in the market, to the

egg-rolls, where, together with the steamboat-size, it is broken down to egg and smaller sizes, which are separated by a special

PLATE IX.



CRANBERRY BREAKER.
DETAIL OF MIDDLE POSTS OF SLOPE PLANE.

screen, the "over"-products of which are egg, stove, chestnut and pea, all of which pass directly to the pockets without jigging. The "through"-product from this screen passes to a

buckwheat- and No. 2 buckwheat-screen, which sizes pass directly to the pockets or to the boiler-house.

At the picking-spaces of the steamboat- and broken-coal the "bony" coal, consisting of pieces that are part slate and part coal, is picked out and passed to a set of bone-rolls. There is one set of bone-rolls and one set of egg-rolls for each side of the breaker. The product from these bone-rolls is divided and passes into the chestnut-screens.

Returning now to the "through"-product of the steamboat- and broken-screen, it will be followed to the pockets. It is at once passed to a second screen, the "over"-products of which are egg and stove, which pass to jigs, then to picking-spaces, and then to the pockets. The "through"-product of this egg- and stove-screen passes to a third screen, the "over"-products of which are chestnut and pea, which pass to jigs and then directly to the pockets. The "through"-product of the chestnut- and pea-screen passes to a fourth screen, the "over"-products of which are buckwheat and No. 2 buckwheat, which pass directly to the pockets or to the boiler-house. The "through"-product of the buckwheat- and No. 2 buckwheat-screen passes to the settling-tanks, with the "wash-water."

From the pockets the prepared coal is drawn into cars for shipment. The pockets, eleven in all, have a storage-capacity of 100 tons each, to insure the running of the breaker against any shortage in cars.

The coal, from the time it is dumped from the mine-cars at the top of the breaker until it reaches the cars for shipment, is made to pass from one process to the next by gravity only, no elevators being used. To accomplish this it was necessary to build the structure to the unprecedented and apparently dangerous height of 200 feet,—apparently dangerous because of the wind-pressure and the constant vibrations from the heavy machinery placed so high above the ground.

The details of this relative arrangement of the machinery, as worked out and constructed, are shown in elevation and cross-section on Plates II. and III., and in plan on Plates XV. and XVI. Plate XV. shows the crusher- and prepared-rolls, and Plate XVI. shows the egg-rolls, bone-rolls and screens, with their driving-gears.

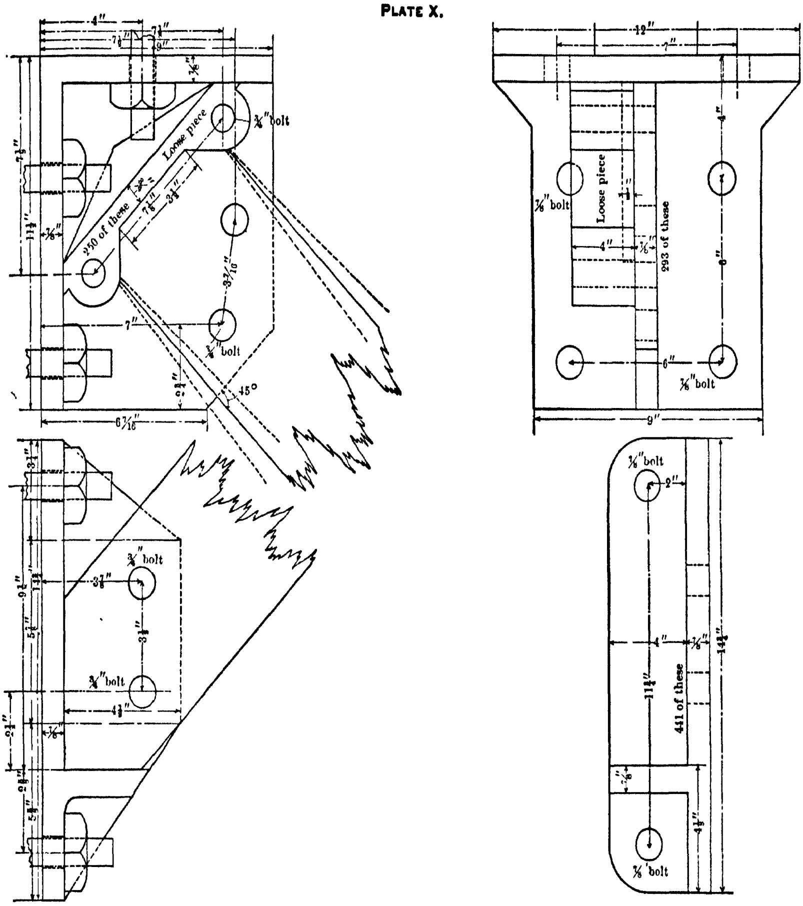
The capacity of this machinery is 2000 or more tons of merchantable coal per day of ten hours.

A brief summary of all the machines is as follows, viz.:

Rolls.—1 pair crusher-rolls, 36- by 72-inch shell, fitted with 286 steel teeth in each roll.

2 pairs prepared-rolls, 24- by 48-inch shell, fitted with 280 steel teeth in each roll.

PLATE X.



BRACKET AND BRACE CASTINGS, CRANBERRY BREAKER.

2 pairs egg-rolls, 24- by 36-inch shell, fitted with 520 steel teeth in each roll.

2 pairs bone-rolls, 24- by 36-inch shell, fitted with 2700 steel teeth in each roll.

Screens.—All screens are of the revolving type.

4 steamboat- and broken-screens, 12 feet long, with two rings of segments, one 5 feet in diameter and the other 7 feet.

4 egg- and stove-screens, of the same dimensions.

4 chestnut- and pea-screens, of the same dimensions.

6 buckwheat- and No. 2 buckwheat-screens, of the same dimensions.

2 egg-, stove-, chestnut- and pea-screens, 18 feet long, with three rings of segments, 5, 7 and 9 feet in diameter.

Jigs.—4 broken-coal jigs.

4 egg-coal "

4 stove-coal "

4 chestnut-coal "

2 pea-coal "

Before passing to the description of the individual machines, a glance at the dump at the top of the breaker, shown in elevation on Plate II., may be of interest. In order to hoist to the dump, in 2-ton mine-cars, coal to the amount of 2000 tons, or 1000 cars per day, it is necessary to hoist 3 cars at a time. The cars are hitched together with extra-long chains, and the three tracks over the dump are arranged so that the cars, when in position to dump, stand as shown on Plate II., thus making room for the coal dumped from a car above to drop into the dump without striking the car below or lodging on its bumpers. In hoisting three cars at a time, it will be plainly seen that there is three times the pull on the hitching-staple at the upper end of the upper car that would occur in hoisting only one car. This triple strain is increased as the cars are pulled over the corrugated track of the dump, and it acts in an oblique direction on this staple, alternately up and down, causing a bending-action that would be sure to break the ordinary staple, and cause serious run-aways. Those between the cars are acted upon obliquely also, but with less strain. This danger was anticipated and forestalled by replacing the old staples with new drop-forged steel staples, designed by the writer, having brace-wings, as shown in Plate XIX. Thus far, not a single staple has broken under the bending-action of this intense oblique pull.

Of the machines, the rolls and jigs play the most important part.

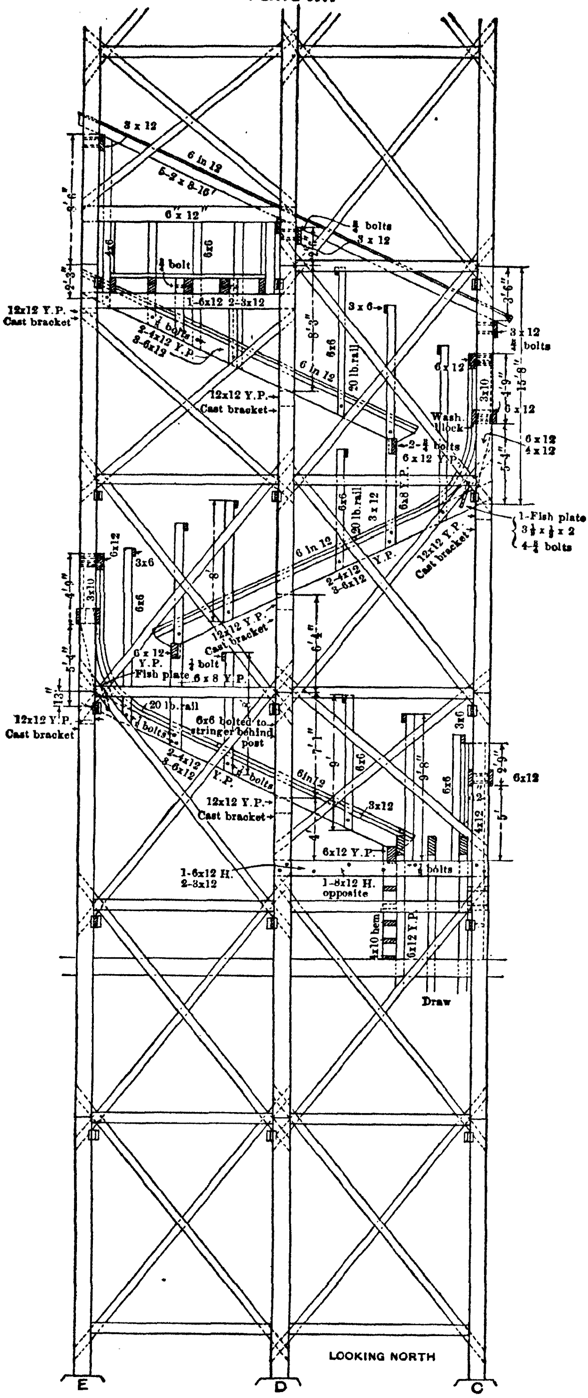
The rolls were increased both in diameter and length over those ordinarily in use, and the teeth were made with the points hooked forward in the direction of the motion. The section of the tooth is like the area included between the segments of two

overlapping circles. This forms a tooth with rounded sides and cutting edges. The tooth also tapers to a point which hooks forward. The rolls of the larger diameter produce a smaller percentage of the small sizes of coal. This is due to the piece of coal being acted upon by the teeth in a way that approximately splits it into pieces. In the old small-diameter rolls, the opening between them widens upward so abruptly that the piece of coal rides on the teeth, and is abraded by them before it is pierced and carried through. This is due to the angle at which the piece is first struck by the teeth.

The screens are of the revolving type, having a greatly increased screening surface. Eighteen of these are double, having an inner circle of segments 5 feet in diameter and an outer one of 7 feet; and all are 12 feet long. The remaining two have a third circle of segments added, 9 feet in diameter, and the 5-foot circle is extended 6 feet, making a total length of 18 feet for the 5-foot circle and 12 feet for the other two.

The jigs, designed and patented by Mr. F. Pardee, are illustrated in Plate XVII. They consist of a water-box 5 feet wide, 9 feet long and 4 feet deep, with a draw-off hopper-bottom. In this box, close to one end, operates an elevator to lift the separated coal out of the water. At the other end is an inclined device that extends into the water and closely approaches the lower end of the coal-elevator. The frame of this inclined device is covered with a series of plates, coupled together with hinges, so as to form a continuous conveyor which travels around the frame, going up on the upper side and down on the lower side. These plates are of different widths for the different sizes of coal, and range from 4⅝ inches to 9¼ inches, from center to center of hinges. They are each provided with a step-like offset, varying in width with the width of the plate, but in length they are uniformly 4 feet, 1¼ inches; hence the surface of the conveyor presents a series of little steps. The total length of each plate, including the part at each end which works in the guides, is uniformly 4 feet, 6¼ inches. This conveyor is provided with a shaft at the lower end of the frame, and one at the upper end, to enable it to travel around the ends. The one at the upper end receives the power to move the conveyor. The whole frame, thus equipped with the conveyor, is hung so as to oscillate, fully half of its length being under

PLATE XI.

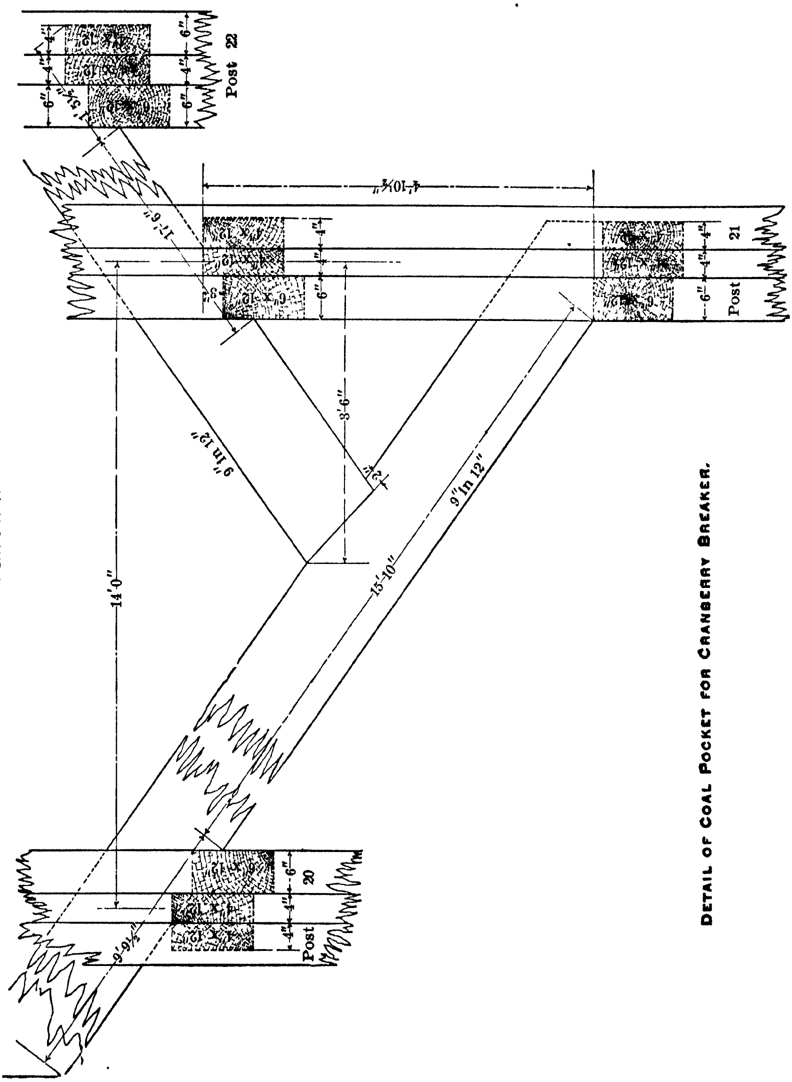


CRANBERRY BREAKER.

DETAIL OF ROCK CHUTE BETWEEN BENTS 17 AND 18.

water. The lower end is hung by two hangers, while the upper end rests upon two rocker-arms that have one end slipped over the upper conveyor-shaft and the other keyed to a rocker-shaft. The oscillating motion is communicated to the conveyor by means

PLATE XII.



DETAIL OF COAL POCKET FOR CRANBERRY BREAKER.

of two eccentrics on the main driving-shaft, one on each side of the water-box. The side of the coal-elevator facing the conveyor travels up; the conveyor travels up also; and the unjigged coal is fed upon the travelling and oscillating conveyor-

plates at a point about 12 to 16 inches below where they emerge from the water. The pure coal is floated away from the steps by each oscillation, and finally falls into the elevator, while the slate, owing to its greater specific gravity or flat shape, remains on the steps and is conveyed out of the water and to the top of the conveyor, where it falls off into the slate-chute.

The jigs are driven by small engines, one for each jig.

As already stated, there are only eighteen of these jigs in the entire breaker, and, when running full, 2000 tons of coal per day of ten hours can be prepared with them.

The approximate average capacity of these jigs per day of ten hours is as follows:

	Tons per day.
Broken-coal, each jig,	250
Egg-coal, "	225
Stove-coal, "	175
Chestnut-coal, "	150
Pea-coal, "	150

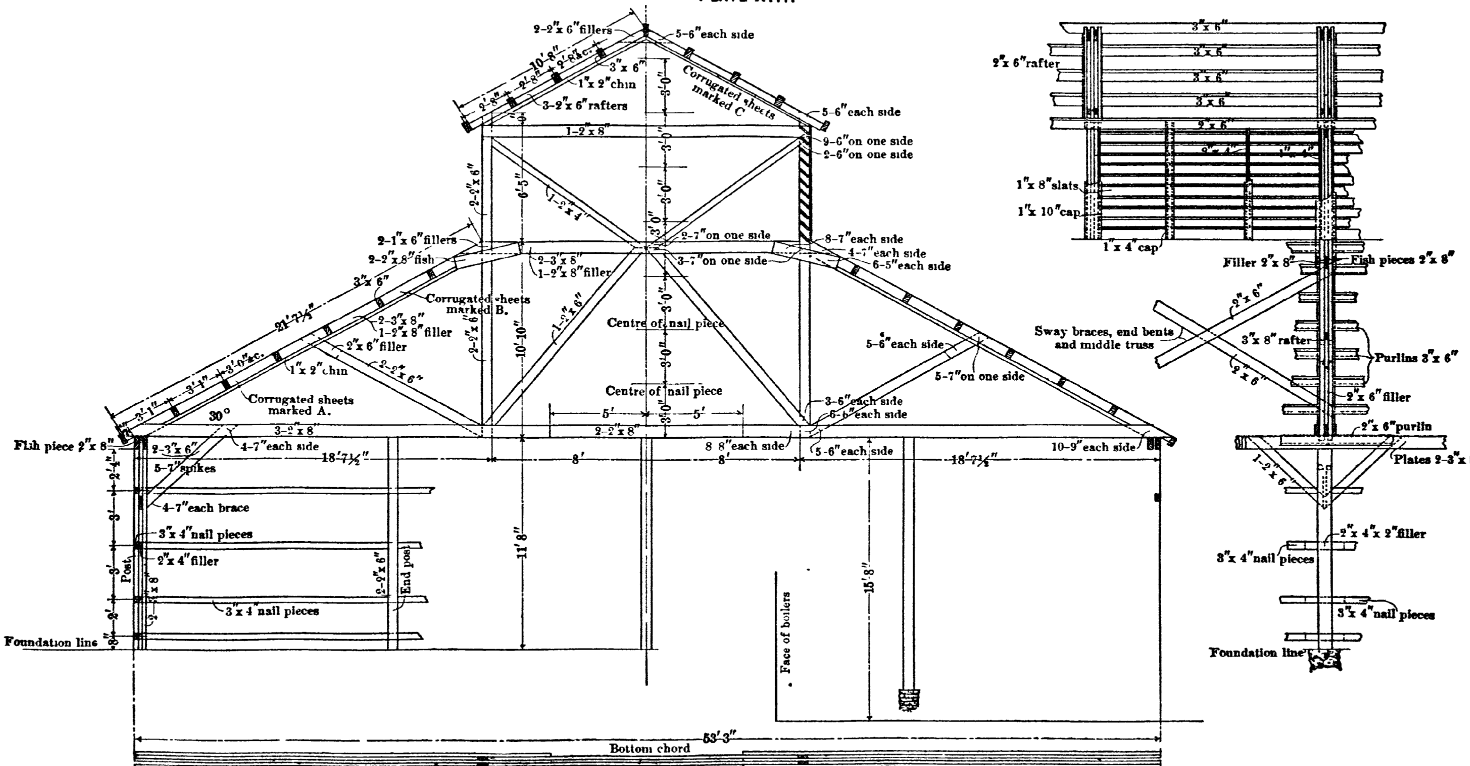
As is well known, all coal is subjected to a rigid inspection as to the percentage of slate and bone it contains, before leaving the breaker. The percentage of coal going over in the tails or slate from these jigs is about 4 to 9 per cent. from stove-coal jigs, and 4 per cent. from pea-coal jigs.

It is worthy of mention here that Mr. Pardee has conducted a very wide range of experiments with this jig, covering numerous combinations of speed, pitch of conveyor, shape of plates, method of hanging, length of waves produced by the oscillations and methods of feeding, the results of which are shown in Plate XVII.

The entire machinery of the breaker is driven by a 30- by 48-inch non-condensing engine. The refinements of a quadruple-expansion and condensing-engine do not receive much consideration where there is an abundance of waste fuel at hand, as at a coal-mine.

The transmission of power from the engine to the counter-shaft, and from the counter-shaft to the various rolls, is by a manila rope-drive. From the engine to the counter-shaft ten 1½-inch ropes are used; from the counter-shaft to the crusher-rolls, four 1½-inch ropes; from the counter-shaft to the prepared-rolls, two 1½-inch ropes; and from the counter-shaft to the bone-rolls, two 1½-inch ropes. The egg-rolls are driven by

PLATE XIII.



ROOF-TRUSS OF CYLINDER BOILER HOUSE, CRANBERRY BREAKER.

AM. BANK NOTE CO N.Y.

gears from the bone-roll shaft. This arrangement of the rope-drive is shown in elevation by Plate II. and in plan by Plate XVIII.

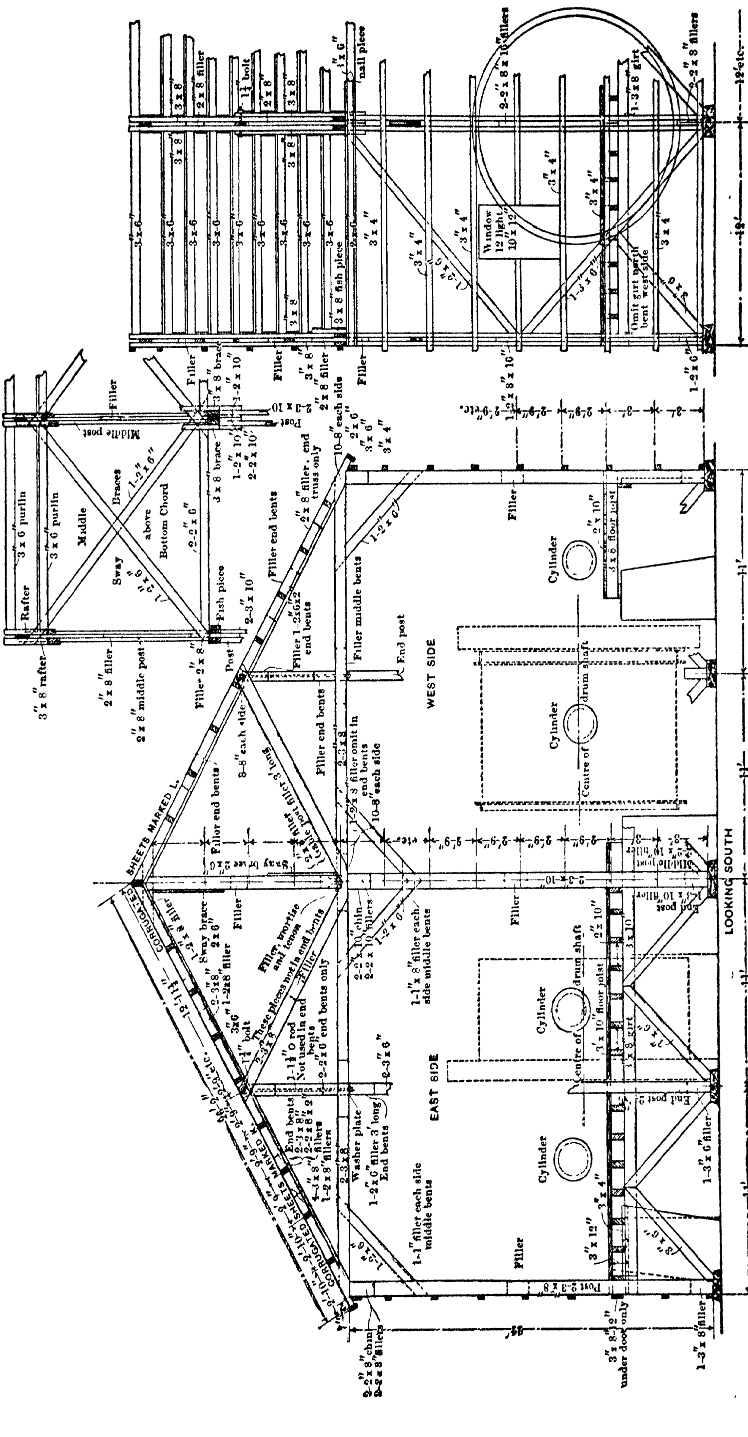
The entire counter-shaft, and in fact the entire rope-drive, except the main driving-sheave at the engine and the sheaves at the rolls, is out of doors and without any protection from the weather. The weather seems to have some effect upon the ropes to tighten them in wet weather and to make them more slack in dry weather, but the range of this change is not sufficient to give any trouble, provided the ropes are adjusted to the right tension when dry. The first set of ropes was put on in September, 1896, including only five ropes on the main drive from the engine to the counter-shaft, and also only those from the counter to the east-side machinery of the breaker. On July 1, 1897, the west side was put in operation, and the remaining ropes were put on. All of these ropes are still in operation without renewal.

Many interesting problems were discussed in connection with the selection of a means of transmission of the power of the engine. The counter-shaft is about 85 feet above the engine-shaft, and so nearly directly over it that it was impossible to count upon any grip on the pulley or sheaves being furnished by the weight of the slack-side of either a belt, a wire rope, or a hemp rope. Therefore, it was necessary to adopt some convenient way to take up all slack and insure a perfect grip.

A leather belt was discarded, because it would be very difficult to keep it at the proper tension without a tightening-pulley, which is a very objectionable device, and also because it would require housing, besides costing about five times as much as a manila rope-drive.

A wire rope was found to be uncertain in its grip on the sheaves, and its life could not be warranted. To secure the required grip, a continuous-rope wound system would be required, with a take-up for wear and stretch, which in turn would require some system of differentiation in the grooves of the drum, like the Walker differential drum, to prevent one round of the rope from running away from the others and doing all the work or breaking, a result which may follow a very slight difference in the diameter of the grooves or of the rope. A wire rope-drive was, therefore, out of the question, because of its complications, uncertainty and cost.

PLATE XIV.



ROOF-TRUSS OF HOUSE FOR HOISTING ENGINES, CRANBERRY BREAKER.

AMERICAN BANK NOTE CO. N.Y.

In like manner the manila continuous-rope wound system was considered and discarded, in view of the objection that if the rope should part for any reason the whole system would be unwound and serious delays would occur in rewinding it.

In the manila individual-rope system were found the advantages of a very low first cost, non-requirement of housing, and ability, if one rope should part, to continue work at once with the remaining ropes.

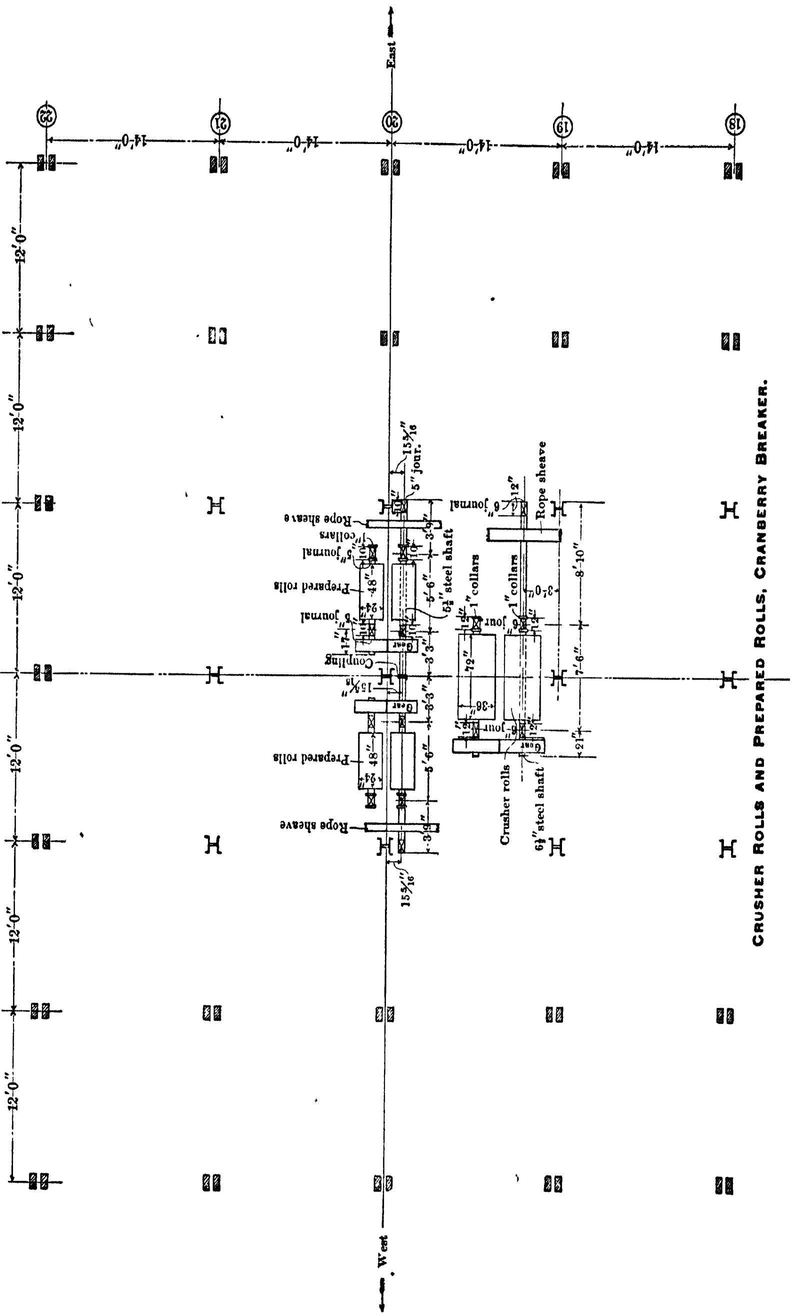
The question of differentiation in the grooves in the manila rope-drive, although the necessity of it exists, is not as serious a problem as in the wire rope-drive, owing to the greater elasticity of the manila rope and to the V-shape of the grooves used. Yet the question arose, How would a new rope work with an old one that is much reduced in diameter by wear? Would the V-shaped groove accommodate this great difference in the diameter of the ropes? It was feared not. The difficulty was solved, however, by the adoption of a shackle and coupling-link like that used by the Philadelphia & Reading Coal and Iron Co. at its storage-docks at Schuylkill Haven, Pa. This shackle is of the ordinary U-shape, is made very short, and has the legs flattened into very thin plates, so as not to increase the diameter of the rope very much. A shackle is riveted on each end of the rope, and the two are hooked together by an open link. This slightly increased diameter at the shackle lifts the rope out of the groove a little as it passes over each sheave, enabling the rope to yield a little toward the side that is drawn too tight. This solved the difficulty of differentiation entirely.

The ropes are shortened, as wear and stretch make it necessary, in two ways. One is to unhook the coupling-link and put more twist in the rope, and the other is to cut off a piece of the rope at one end, and replace the shackle. This is done in the noon-hour, or after-time, but is not a matter of very frequent occurrence after a new rope has received its permanent set.

The number of ropes necessary to drive the rolls was doubled, to provide against any delays from the parting of a rope. On the main drive several were added for the same reason.

The diameter of the sheave on the engine-shaft is 20 feet, with 10 grooves in its 22-inch face. On the counter-shaft the main sheave is 10 feet in diameter, while all others are 6 feet.

PLATE XV.



CRUSHER ROLLS AND PREPARED ROLLS, CRANBERRY BREAKER.

The speed of the engine ranges from 50 to 60 revolutions per minute, as desired. The speeds of the ropes on the main

sheave, corresponding to these speeds of engine, are 3100 to 3800 feet per minute.

The screen counter-shaft, shown in plan on Plate XVIII., is driven from the main counter-shaft by bevel-gears, and in turn drives the eighteen 12-foot screens by means of link-belts. These link-belts have a speed varying with that of the engine from 470 to 565 feet per minute, and work exceptionally well as a drive in a dusty, dirty place.

All the machinery was very carefully planned and worked out in every detail, with special regard for convenience in making repairs. As a result, it all went together without any difficulty, and worked from the start.

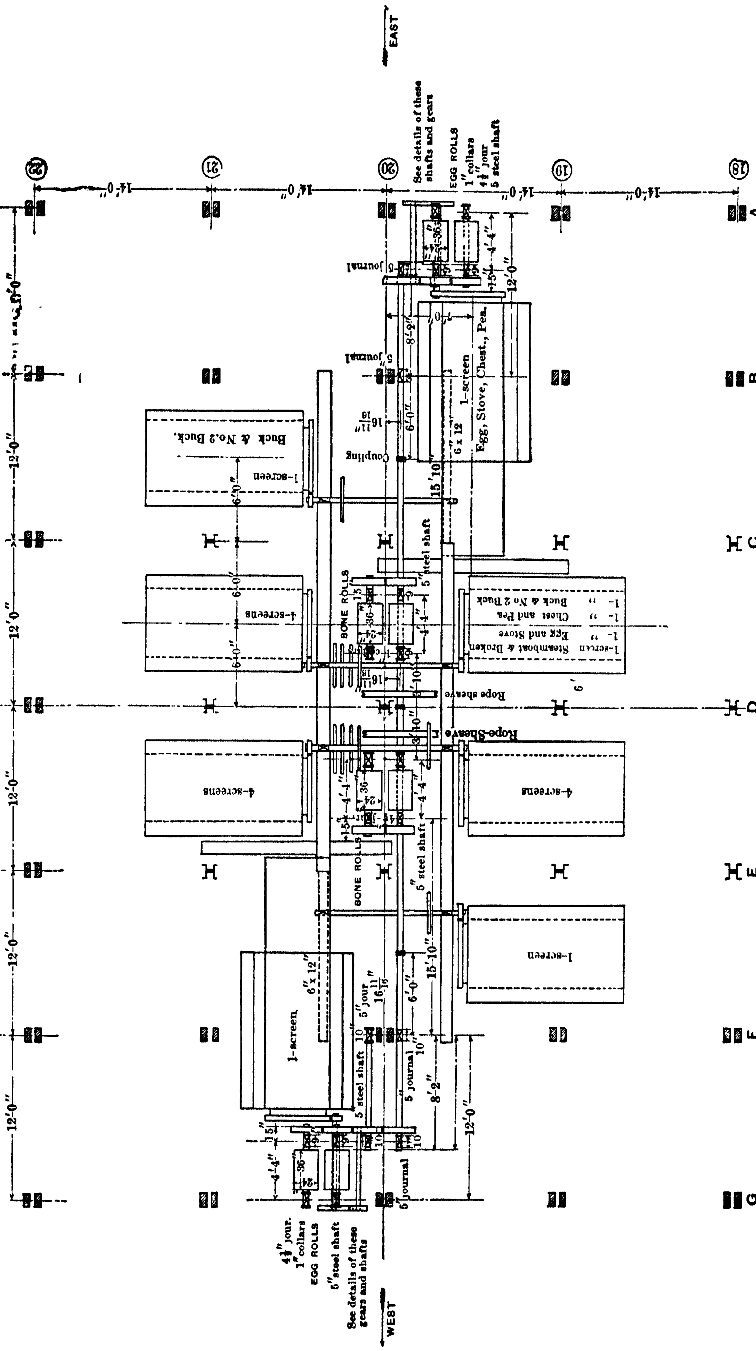
Chutes and Connections.

Owing to the great height of the breaker, it was necessary to resort to some special and unusual devices for delivering the slate and rock, and some of the coal, from points high up in the breaker to the loading tracks below. Among these will be mentioned only the device in the rock-chute between bents 17 and 18, the spiral chutes from the jigs to the pockets and the spiral lump-coal chute.

The usual way of delivering the waste and the coal to the loading-tracks has been by long, straight chutes, down which the slate or coal runs, at times with a cannon-ball velocity. Turns have frequently been introduced, so as to avoid certain spaces or to gain more distance in which to accomplish the descent. This system requires a great deal of lumber, and occupies a great deal of valuable space, besides having the objection as to the coal-chutes, of chipping the coal badly at every turn, and also where it drops into the pocket. These chippings are small in size and worthless, but their constant production, even in small quantities, reduces in a year, by many tons, the output of the larger sizes of coal, which are most valuable. In the slate- and rock-chutes, however, no such considerations arise, and they usually take the shortest route. One necessity, however, does arise, namely, to provide against the impact of the large pieces at the turns or at the draw. This is usually done by making the timbers at the turns very heavy.

On Plate II., in the rock-dump, where it turns at the top of bent 12, is shown the device used to avoid the full impact of

PLATE XVI.



EGG ROLLS, BONE ROLLS AND SCREENS, CRANBERRY BREAKER.

the large pieces, or of a car-load of mixed sizes, dumped suddenly and moving as one mass. Each car of rock dumped

into this chute weighs approximately from 3 to 3½ tons. This device consists of a vertical return, in which the material dumped on the upper section is delivered at an oblique angle against the curved end of the lower section. The force of impact in this device is only about two-thirds of that produced when the piece is brought against the turn at a right angle. On account of this condition, less timber was used to provide against impact.

This rock-chute down to bent 17 is perfectly straight, but between bents 17 and 18 it makes two horizontal and two vertical turns. These are shown on Plate XI. as a "detail of rock-chute between bents 17 and 18." The bottom of the chute at each turn is completely covered with 20-pound rails, laid with the heads up, as in a track, and as close together as the bottom flanges will permit. They are slotted in the flanges, and fastened down with bolts. Such a bottom presents a series of rail-heads to receive the wear, and they are separated about 1¼ to 1½ inches.

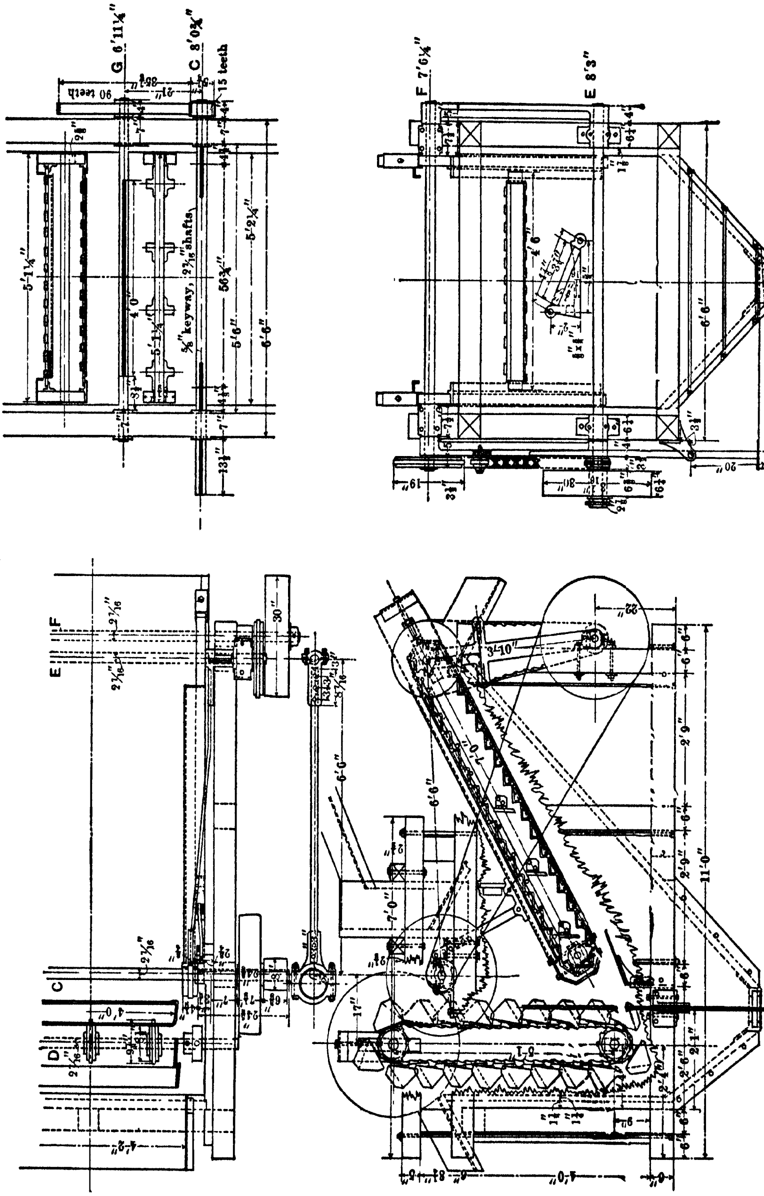
When a piece of rock slides down the long straight part of the chute and enters the horizontal turn it strikes the heads of these rails a glancing blow across their length, and the greater part of its force is expended in passing from rail to rail. Thus the full force of impact is avoided in the horizontal turns. At the lower part of the chute, where the rock and slate are drawn into the dump-cars, there is another horizontal turn, constructed in the same manner. Between these two horizontal turns will be seen, on Plate XI., two vertical turns which operate in the same manner as the one in the dump at the top of bent 12, above described.

Thus a drop of 82 feet is made in a horizontal distance of 72 feet, and without any destructive shocks.

From the jigs to the pockets, the vertical drop ranges from 70 feet from the broken-coal jigs to 34 feet from the pea-coal jigs. To convey the coal from the jigs by the usual inclined troughs, or "telegraphs" (a good name for this trough, for the coal often goes through it at almost lightning-speed), would require many thousand feet of lumber, besides which the building would need to be increased by about one-half of its present floor-space to accommodate them. About a mile of these "telegraphs" would have been required. After discussing this

method, together with several modifications of it, the possibilities of a spiral chute, or, more correctly speaking, a helicoidal chute, were brought forward and carefully gone over, with the

PLATE XVII.



F. PARDEE, COAL JIG.

result that, after some hesitation, they were adopted. To determine the correct pitch, so that the coal would neither stop nor run too fast in the chute, was a problem concerning which

there were no practical data to be found. From the experience with quarter-turns, both in the old breaker, and also in other breakers in this region, it was decided to adopt a pitch of 3¼ inches per foot for the outside circle.

A chute was then constructed as follows:

The outside diameter is 36 inches and the inside diameter 16 inches. The cross-section is uniform for all sizes of coal from broken to No. 2 buckwheat, the width being 10 inches, and the heights of the sides or flanges 4 inches for the outside and 2½ inches for the inside. The chute is made of cast-iron, with four segments to the circle. They are also made rights and lefts. Each segment is provided with four holes in its outer flange, two at each end, and all are countersunk on the inner side, and are large enough to take an ordinary ⅜-inch chute-nail. These segments, or spiral chute-plates, are nailed between four upright posts of yellow pine each 4 by 6 inches, and each plate joins the next one at the center of a post. No support is used for the inside edge of the plates. The surface of each plate is made to match that of the one below by a beveled joint, so made that the plate above rests on the one below.

The floor-space occupied by these chutes is very small, being only about 3 feet square for each chute, and no difficulty occurred in finding space for them.

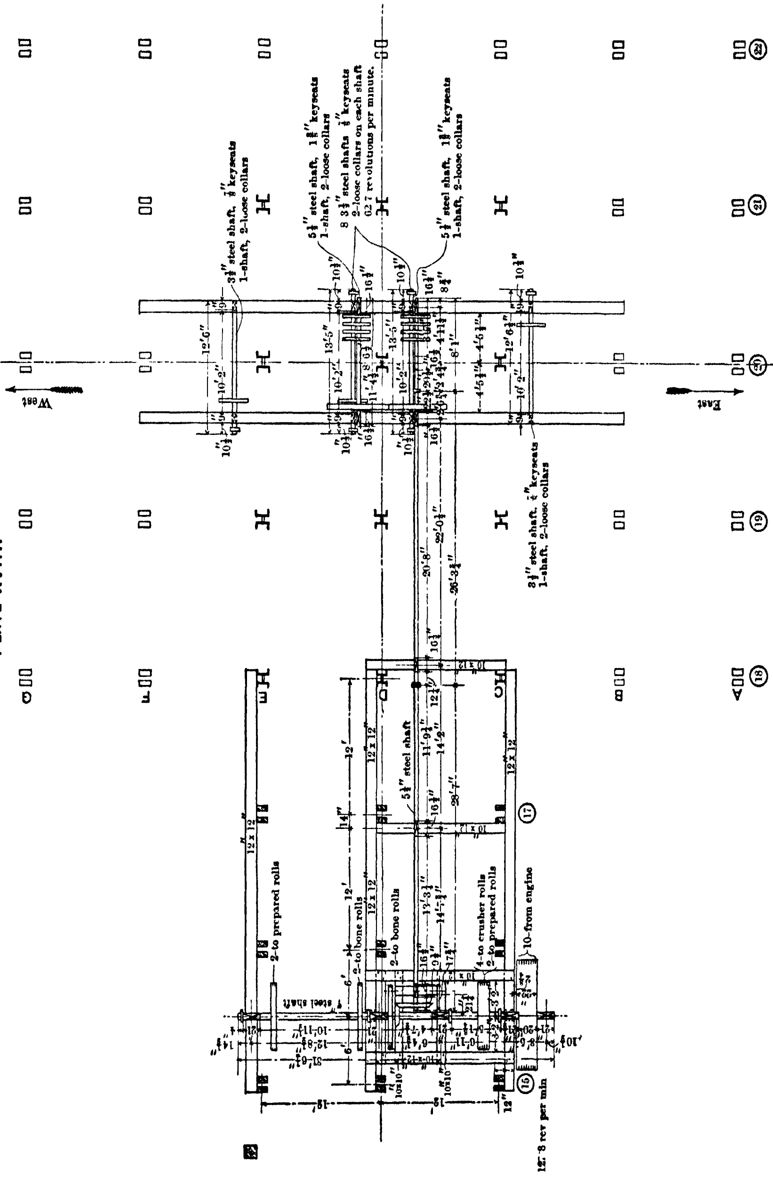
In the practical operation of this chute it was found that the pitch was not great enough for the smaller sizes of coal, and that the chutes would choke. This was remedied by taking the plates out, and nailing them in again in a tilted position, which gave them more pitch. Since this change, no difficulty has been experienced.

When the machinery was put into the west side of the breaker, from April to June, 1897, a new set of chutes was made, having a different pitch for each size of coal, as determined by the experience on the east side. Only the pitch was changed,—all other dimensions were retained.

Between a jig and the place selected for its spiral to go down, a connecting-chute is made of these spiral plates by using alternately a right and left segment. They are nailed to a 12-inch plank that has been stiffened by a backing-piece, 6 inches wide, set edgewise and spiked to its under side. These right and left

plates form a zigzag chute, through which the coal passes without any acceleration. When the coal enters the spiral it at once crowds against the outside flange, and the pieces tend to

PLATE XVIII.



COUNTER SHAFTING, CRANBERRY BREAKER

arrange themselves in a tandem order, forming, particularly in the larger sizes, a continuous stream of single pieces of coal. When the speed increases above a certain amount, each piece

leaves the bottom of the plate and runs against the outside flange, where it assumes a uniform speed which is regulated by the centrifugal force pressing it against the side and causing greater or less friction, according to the tendency of the piece to accelerate or retard.

In the pockets the spirals run to the lowest points desired, so that all the coal will be delivered into them without any drop or impact. They are made without any inside flange, and the action of the coal is to slide off the inside edge of the spiral at the immediate surface of the conical mass that is accumulating in the pocket, and to slide down and over its surface. No impact of any consequence occurs from the time the coal leaves the jig until it is at rest in the pocket, and the amount of chippings, therefore, is very small.

It is fascinating to see the coal *en route* to the pockets. First it goes like a snake and then takes a spin.

The lump-coal chute, which begins at the bars over the crusher-rolls, makes a drop of 110 feet in 78 feet horizontal. This chute has two sections which are also constructed as spirals. It is shown, in elevation and detail, on Plate III., at the right or east side. The same want of data, as to pitch, confronted us in the construction of this chute as in the construction of those just described. It should be clearly understood that the pitch of a spiral does not follow the same law as that of a straight chute. Under these circumstances, and in the anxiety to avoid, particularly in this chute, the usual cannon-ball velocity, the same error was made, as to the pitch, that was made in the others. The pitch proved to be not quite great enough. It has not yet been changed to a greater pitch, because lump-coal is not in demand and therefore is not now made. There is no doubt, however, that it will work.

Results.

Foundations.—The foundations are carrying their respective loads without any settling. This is due to the fact that the intensities of pressure assumed to be safely carried by their various parts are, in practice, within the limits consistent with the stability of the earth upon which they rest, and of the safe working-pressures for the stone and cement used.

Frame of Structure.—In the plane, the advantages gained by

the use of the open-member, narrow-base construction, stayed with guy-lines, are briefly as follows:

1. The narrow-base avoids occupying valuable room wanted for other purposes, that would be fully occupied by any self-sustaining broad-base construction.

2. The guy-lines hold the top of the structure perfectly rigid, while in any self-sustaining structure the sway will increase with use.

3. The open members have all the advantages over the solid members, which have already been mentioned in the description of the frame. Prominent among them are:

a. Greater rigidity of the frame.

b. The commercial sizes used are cheaper and always obtainable in emergency.

c. They are more easily inspected.

d. They will season better and dry out quicker after being wet, and therefore are less liable to rot.

e. Each piece is easily removed and replaced in repairing.

f. The open-work in the columns, ties, and cross-ties, and the use of braces acting in tension only, consumes generally about two-thirds of the lumber, for the same strength, that is required by the solid-member construction.

In the plane as constructed, about 127,000 feet of lumber were used, 24,329 pounds of bolts and washers, and 5972 feet of $1\frac{1}{8}$-, $1\frac{3}{16}$- and $1\frac{1}{4}$-inch guy-lines.

The saving in lumber as compared with a broad-base self-sustaining structure, built after the same open-member system, is about 100,000 feet. And as compared with a broad-base solid-member system it is about 200,000 feet. The value of this timber at $25 per thousand, erected, is $5,000. Against this must be charged the guy-lines and their connections (about $800), and the bolts and washers (about $550), or $1350 in all, leaving as a net saving to the credit of the method used, $3650, in the plane alone.

In the breaker a similar saving of about 250,000 feet was effected, which, added to that saved in the plane, makes a total saved in the entire structure of 450,000 feet of timber. The value of this 250,000 feet at $30 per thousand, erected, is $7500, against which must be charged 45,951 pounds of bolts and washers (about $1100), leaving, as a net saving, in the

break , $6400. This, added to the saving in the plane, gives, as the total net saving in the entire structure, to the credit of the method used, $10,050.

The old breaker, which was a smaller structure, contained over 1,300,000 feet, B. M., and the new one just described contains about 900,000 feet, including the equivalent of the steel columns and exclusive of the trestles to the slate-banks.

The lumber was furnished by the operator, and the contractor framed and erected it at a given price per thousand feet, B. M.

The contract was awarded to S. D. Kingsley, of Olyphant, Pa., and the entire erection was performed without an accident of any kind.

The number of skilled carpenters required was comparatively very small. Instead of an army of skilled carpenters at work making mortises and tenons, only a few skilled men were required to direct the assembling of the pieces, to lay off their proper distances, and to mark the positions of the bolt-holes.

All of the contractors bidding on the framing and erection were very wary, and even fearful that the structure would not stand up. In fact, some were very pronounced in their disapproval of it. But such views are not now entertained.

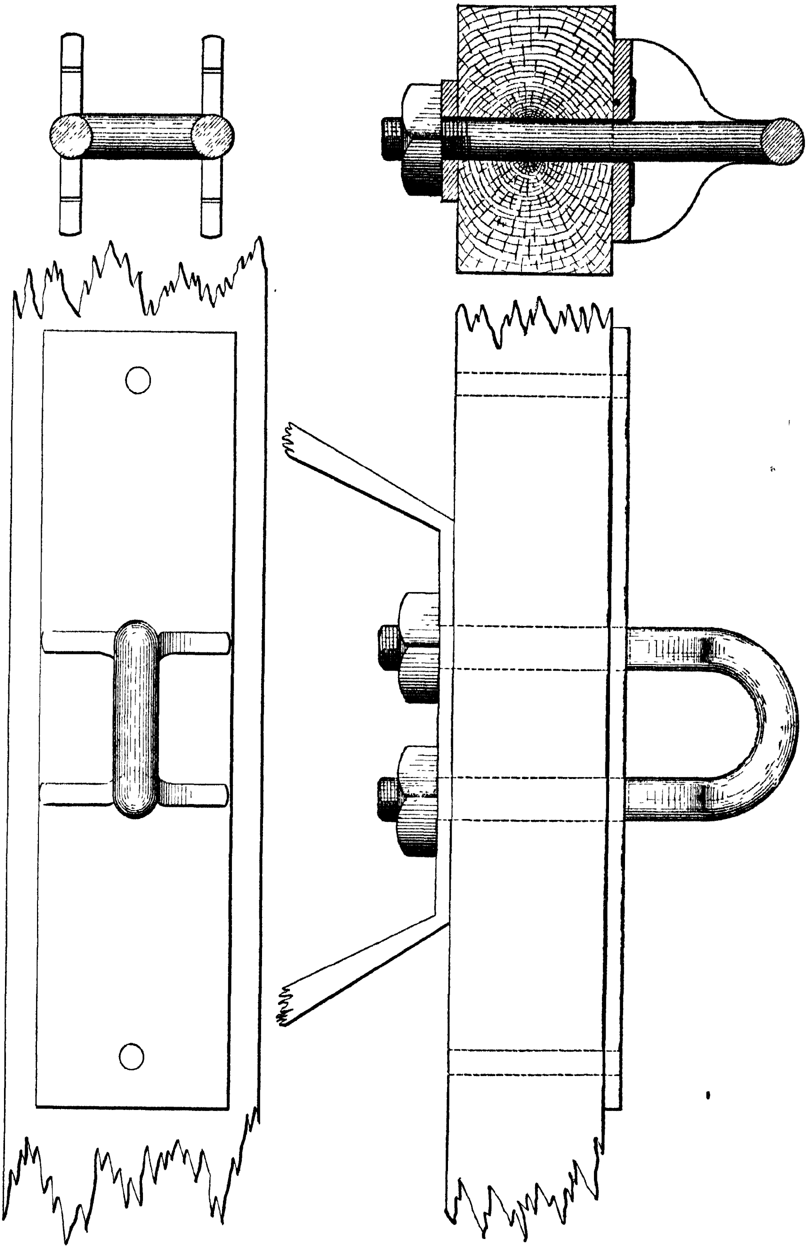
Machinery, Chutes and Connections.—All that will be necessary to give here as to the efficiency of the machinery, chutes and connections will be the approximate gain effected by making a greater percentage of the larger sizes of coal and a correspondingly smaller percentage of the smaller sizes. Compared with the old breaker, it is estimated that from 10,000 to 20,000 tons, on a yearly output of 300,000 tons, is transferred from pea, buckwheat and No. 2 buckwheat to egg, stove and chestnut.

In this comparison it is also necessary to bear in mind that egg, stove and chestnut are the sizes that command the highest prices in the market, while pea, buckwheat and No. 2 buckwheat command the lowest, and lump, steamboat and broken the medium prices.

The above gain is accomplished by the style of the rolls, the simplicity of the machinery, the absence of all elevators, and the avoidance, by the use of the spiral chutes, of any impact.

The item of saving in labor in the cost of preparation should

PLATE XIX.



New Hitching-Staple for Mine-Cars, Cranberry Breaker.
Forged by The Whitman and Barnes Mfg. Co., Akron, Ohio.

here be mentioned, as it also stands to the credit of the construction of the breaker and of the efficiency of the machinery.

In the old breaker, to prepare 1150 tons per day, 194 men and boys, on an average, were required. In the new breaker, to prepare 2000 or more tons per day, only 84 on an average are required. Mr. Pardee has recently made some important changes in his jig that will in all probability reduce the number of slate-pickers from 55 to not more than 25, and they will be employed only on coal that is not jigged. This will further reduce the total number of men required from 84 to 54.

As to the capacity of the breaker, it has been intimated, under the head of "Machinery," that more than 2000 tons of merchantable coal could be prepared in a day of ten hours. All calculations and expectations in this direction have been more than realized, and it is now assured that 3000 tons can be prepared instead of 2000.

The tendency in the preparation of anthracite coal has been, in recent years, to complicate rather than simplify the process, particularly since the late Eckley B. Coxe read before the Institute his famous paper on the "Iron Breaker at Drifton," in which he described so many ingenious refinements in coal-preparing machinery.

In the construction of the Cranberry breaker the greatest simplicity was aimed at, so as to avoid the frequent handling and consequent undue chipping of the coal, the costly repairs on complicated and expensive machinery, and, not least of all, to do the work of preparation with a far smaller number of men.

I desire, in closing this paper, to acknowledge the courtesy of Mr. F. Pardee, Superintendent, in freely allowing the use of all data desired to set forth the facts here given; and I wish also to express my appreciation of the skilful services of my assistant, Mr. J. E. Anderson, who faithfully assisted me in the designing and construction of the breaker.

INDEX OF THE PLATES.

Mining and the Forest Reserves.

BY GIFFORD PINCHOT, WASHINGTON, D. C.

(Atlantic City Meeting, February, 1898.)

THE proposition that forest reserves are, from the mining point of view, not only desirable but necessary, is not one which would have received enthusiastic support in the West a year ago. It is, nevertheless, easily susceptible of proof. A short and simple statement of the facts has seldom failed to carry conviction as to the wisdom of the forest-reservation policy.

Hostility to mining interests was never contemplated by the men to whom the forest reserves are chiefly due, and it is most unfortunate that the opposite belief should have obtained wide

credence in the regions where it was most likely to do harm. A glance at the origin and contents of the rules and regulations which have been most bitterly opposed by mining men will make this clear.

When Secretary Hoke Smith, under the last Cleveland administration, called on the National Academy of Sciences for an investigation of the forests of the country and the preparation of a plan for their management, the President of the Academy responded by the appointment of a commission of seven men. Among the seven were four or five whose acquaintance with the forests west of the Missouri would have been exceedingly hard to duplicate. These men, familiar with the West for thirty or forty years, had traversed it in all directions, both as explorers and as students of its various resources, and were thoroughly equipped for the work before them. In order to refresh their memory of certain portions of the western forest, and to examine regions not yet visited, five members of the Commission spent the field-season of 1896 in the West, and returned at the end of it to make their recommendations. In accordance therewith President Cleveland proclaimed, on Washington's Birthday, 1897, thirteen additional forest reserves, with an aggregate area of more than twenty-one million acres. The form of the proclamation was unfortunate, for, on the face of it, it forbade the prosecution of any industries within the regions affected, and warned all men to keep out. The real intention was something very different. At the time the proclamation was made work was being rapidly pushed on the preparation of a plan of management for the forest reserves, under which the fullest and most continuous development of all the resources of each of these regions was the first and essential object. From the beginning of the work until now this has been the keynote of the friends of right forest-management. Use, equally opposed to abuse and to the Chinese Wall theory of forest reserves, has been the special object of their endeavor.

The following extracts from the Rules and Regulations of June 30, 1897,* prepared for the government of the reserves and now in force, will explain better than any general descrip-

* A later edition contains a few changes, but none of general importance.

tion the liberal view now held by the authorities in Washington. Under these regulations, "Public forest-reservations are established to protect and improve the forests for the purpose of securing a permanent supply of timber for the people and insuring conditions favorable to continuous water-flow." The establishment of reserves is thus removed at once from the domain of controversy. If the reserves already made were called into existence for these reasons (and it is true beyond all doubt that such was the case), there can be no quarrel with the general policy of forest reservation.

One of the first and most potent sources of complaint was the fear that access to the reserves would be denied. The law of June 4, 1897, sets this matter permanently at rest. It provides that "Nothing herein shall be construed as prohibiting the egress or ingress of actual settlers residing within the boundaries of such reservations, or from crossing the same to and from their property or homes; and such wagon-roads and other improvements may be constructed thereon as may be necessary to reach their homes and to utilize their property under such rules and regulations as may be prescribed by the Secretary of the Interior. Nor shall anything herein prohibit any person from entering upon such forest-reservations for all proper and lawful purposes, including that of prospecting, locating and developing the mineral resources thereof; Provided, that such persons comply with the rules and regulations covering such forest-reservations."

The law goes on to say that "All water on such reservations may be used for domestic, mining, milling or irrigation purposes, under the laws of the State wherein such forest-reservations are situated, or under the laws of the United States and the rules and regulations established thereunder."

This last clause is interpreted by the department to cover rights of way under existing laws and regulations. The resources of the reserves in water are therefore as fully open to use as those of unreserved lands, and in addition the matter of permanence of water-supply gives the reserves a distinct advantage.

The provisions of the laws and the regulations with regard to mineral lands are of the first importance. To quote the regulation:

"The law provides that 'any mineral lands in any forest-reservation, which have been or which may be shown to be such, and subject to entry under the existing mining laws of the United States, and the rules and regulations applying thereto, shall continue to be subject to such location and entry,' notwithstanding the reservation. This makes mineral lands in the forest reserves subject to location and entry under the general mining laws in the usual manner.

"Owners of valid mining locations, made and held in good faith, under the mining laws of the United States and the regulations thereunder, are authorized and permitted to fell and remove from such mining claims any timber growing thereon, for actual mining purposes in connection with the particular claim from which the timber is felled or removed."

The section of the act which allows the free use of timber and stone by bona fide settlers, miners, residents and prospectors, and its interpretation by the department, are as follows:

"'The Secretary of the Interior may permit, under regulations to be prescribed by him, the use of timber and stone found upon such reservations, free of charge, by bona fide settlers, miners, residents and prospectors for minerals, for firewood, fencing, buildings, mining, prospecting, and other domestic purposes, as may be needed by such persons for such purposes; such timber to be used within the State or Territory, respectively, where such reservations may be located.'

"This provision is limited to persons resident in forest-reservations who have not a sufficient supply of timber or stone on their own claims or lands for the purposes enumerated, or for necessary use in developing the mineral or other natural resources of the lands owned or occupied by them. Such persons, therefore, are permitted to take timber and stone from public lands in the forest-reservations under the terms of the law above quoted, strictly for their individual use on their own claims or lands owned or occupied by them, but not for sale or disposal, or use on other lands, or by other persons: Provided, that where the stumpage-value exceeds one hundred dollars, application must be made to and permission given by the Department."

The law does not overlook the fact that large quantities of timber required in mining operations by companies and corpo-

rations cannot be taken under the head of "individual use." Provision to meet this need is amply made. The Secretary of the Interior has power to designate, appraise, and sell practically any merchantable timber in the forest reserves, as he may see fit. Nor is private initiative excluded. It is expressly stated in the regulations that petitions from responsible parties will be considered, and specific directions are given for the drawing up of such requests. These provisions seem to remove all difficulty, from the miner's point of view, with a single exception. Where timber in large quantities has been taken without charge in the past, some share of the cost of caring for and preserving it must hereafter be borne by the men who benefit by such protection.

To sum up the whole matter, the law of June 4, and the regulations prescribed under it, may be said to have the following effect on the forest reserves: They establish the purposes of the forest reserve policy on a basis whose wisdom cannot be questioned; they permit access to the reserves by all persons whose legitimate business takes them there, and among such persons prospectors and miners are specifically mentioned; they permit the development of all mineral claims, and the cutting of timber on these claims for use in connection with them; they give without charge timber to the value of one hundred dollars on the stump to prospectors and miners whose claims do not furnish sufficient material for their own use, and they provide for the sale of timber in large quantities to meet the demands of larger operations. The careful study of these provisions will, in my opinion, leave not a peg on which to hang an objection to the forest reserves on the part of men interested in the development of the regions in which they lie; if any such objection can be raised, it must come because of the extreme liberality of the regulations, not in default of it.

These are the reasons why the reservation of any forest land does not interfere with the development of mineral wealth, either within its borders or in its neighborhood. But this is not enough. If the best that can be said for the reserves is that they do not increase the burdens of miners operating in or near them, then but little can be expected of the miners beyond letting them alone. Ineffectiveness on one side will naturally beget indifference on the other. Fortunately the story

is but half told. · The positive value of the reserves to miners everywhere, and especially to those who deal with low-grade ores, is so great that I venture to say that without them many of the enterprises which contribute most to the prosperity of the West would be seriously crippled or made altogether impossible, and that within so few years as to give the whole question the force and pressure of actuality. I do not intend to convey the impression that only the immediate future is worthy of consideration. Even from this standpoint, however, the reserves are justified; but, at the best, this is a narrow view of their value. The willingness to look forward is an essential part of the equipment of any man who is able to judge fairly of any question, either of forestry or of national policy of any other kind. Devotion to the interests of this country which does not consider the elements of growth and permanence is unfair to the nation, in whole and in part.

Briefly stated, the reason of all this is that the forest reserves will protect and continue the timber-supply. They will do this, first, by the prevention of fire, to which cause alone is due the destruction of a much larger quantity of timber than has been consumed by all the industries of the West since the beginning of its industrial development. Fire is the greatest foe to the forests west of the Missouri river. It can be overcome by government action, and by that alone. The forest reserves will be safer from fires than any other portions of the West except the national parks.

The total effect of fire on the value of western forests is a matter as yet but dimly understood. It is easy to grasp the enormous loss which a great forest fire involves when standing in the midst of the dead and bleaching trees, or climbing painfully and slowly over the down timber. This side of the matter is sufficiently forcible to the man on the ground, and it may become so, through him, to others. But the effect of fire on growing timber, and its influence on the forest which has replaced an earlier growth, long since burned away, are matters less easily perceived, but not of less importance. I cannot dwell upon these points in detail, but must restrict myself to calling attention to a well-known fact, that in certain regions of the West timber for mining, or for any other purpose, is scarce and difficult to obtain. To mention only some scat-

tered regions with which I am acquainted, such mining-timber as once existed in the neighborhood of Pike's Peak and Cripple Creek, Colorado, has almost wholly disappeared, on account of fire and injudicious cutting, and has been replaced by worthless species, among which quaking asp is conspicuous. In the Priest River reserve, with its mineral belt in the northwestern portion, forest fires have followed the prospector, although they have not been confined to the regions he has chiefly visited. Fires have burned over almost half the area of this reserve within the last thirty years, and the loss which they have caused to the prosperity and development of the region may, with all possible conservatism, be called enormous. The story elsewhere is nearly the same. In the Washington forest reserve fires have done most serious damage, especially east of the summit of the Cascade range. Near Monte Cristo, on the western slope, the injury already is large and is constantly growing. In the Big Horn mountains, familiar to me only through reports of others, I am informed that not more than 10 per cent. of the whole area is now covered with merchantable timber, and that this result is directly due to forest fires, and to no other cause. Considerable mining development is anticipated in this region by residents, and if it ever takes place, it must depend largely or wholly on indigenous timber. The moral here is very plain. In the Bitter-root forest reserve the development of the mineral wealth west of the main divide has been seriously hindered by the enormous spread of ancient and recent fires. Throughout the mineral regions of the West the same trouble exists. Cutting has done but little harm in comparison with the great damage caused by fire. The government is the only agent capable of attacking this gigantic evil, and even the government is helpless unless it can permanently control the areas with which it must deal. This is the first and most important meaning of forest reservations.

The unnecessary waste of timber under present methods of cutting is another source of loss far greater than is often believed. It is chiefly due to an exaggerated idea of the cost of better treatment. In the Black Hills the preventable loss from this cause is especially severe, and is due to the yearly cutting of thousands of thrifty young trees in the northern hills,

while thousands of cords of material, the use of which would make this cutting needless, are left to rot on the ground a few miles to the south.

Timber in the forest reserves will, then, not be wasted as it is at present. It will be used, but the difference between use and waste in our western forests is something which can be fully appreciated only by men who have seen the thing for themselves. The young growing trees, which are now so often destroyed by fire or by injudicious cutting, and on which the timber-supply in the nearer future almost entirely depends, will be protected, and the reproduction of the forest will be secured, and with it the perpetuation of the timber and the safety and continuance of the industries which depend upon it.

Note on the Use of the Tri-Axial Diagram and Triangular Pyramid for Graphical Illustration.

BY PROF. HENRY M. HOWE, COLUMBIA SCHOOL OF MINES, NEW YORK CITY.

(Atlantic City Meeting, February, 1898.)

THE chief purpose of this note is to call attention to the tri-axial diagram as a convenient means of illustrating the properties of slags, and by this example of its use to commend it to those inclined to graphical studies having other possible applications.

Explanation.—In the common or di-axial diagram, the properties corresponding to any point are determined by its perpendicular distance from each of two axes. For example, in an ordinary stress-strain diagram the properties of a given point in the curve are, first, a stress of so many pounds per square inch, because its *perpendicular* distance from the horizontal axis of the diagram corresponds by the vertical scale to that stress; and second, a strain or elongation of so many per cent., because its *perpendicular* distance from the vertical axis corresponds by the horizontal scale to that elongation.

It happens that, as a matter of convenience, we habitually place these two axes at right angles to each other; so that our measurements happen to be made not only at right-angles to

each axis, which is the essential matter, but also parallel with each axis.

A diagram such as is sketched in Fig. 1, with three axes, OX, OY and XY, gives us the means of expressing three variables, just as the di-axial diagram gives us the means of expressing two. And as, in the di-axial diagram, the properties represented by any point are measured by the perpendicular distance of that point from each of the two axes, so in the tri-axial diagram the properties represented by any point are measured by its perpendicular distance from each of the three axes, OX, OY and XY.

For instance, if these distances represent percentages of silica, lime and alumina respectively, then the point A, Fig. 1, represents 10 per cent. silica, 20 per cent. alumina, and 70 per cent. lime. It represents 10 per cent. of silica because its perpendicular distance from the horizontal axis OX corresponds by the vertical scale to 10 per cent.; 20 per cent. of alumina, because its perpendicular distance from the axis XY corresponds by the alumina scale to 20; and 70 per cent. of lime, because its perpendicular distance from the axis OY corresponds by the lime scale to 70 per cent.*

The use of the tri-axial diagram is limited to cases in which the sum of the three variables is constant, such, for instance, as the percentage composition of ternary compounds, which necessarily add up to the constant sum of 100 per cent. This limitation is indeed serious; yet, after using this form of diagram in my lectures for some fourteen years, I believe it capable of many and important applications.

Let us now take, as an actual case, the properties of an important class of ternary compounds, the silicates of lime and alumina, which form the chief part of our iron blast-furnace slags.

Isocals of the Lime-Alumina Silicates.—The contour-lines in Fig. 1 represent, as inferred from Professor Akerman's very

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* For a fuller description, and for the demonstration of this most ingenious device of Professor R. H. Thurston, see "On a New Method of Planning Researches, etc.," *Proc. Am. Ass'n. Adv. of Science*, 1877, xxvi., p. 114, and paper No. 214, *Trans. Am. Soc. Civ. Eng.*, 1881; Thurston, "The Materials of Engineering," 1884, iii., p. 416; also, Professor J. B. Johnson, "The Materials of Construction," 1897, p. 174.

valuable data,* the "isocals,"† or lines of equal total heat of solidification, of the silicates of lime and alumina.

The information given by these results is the same in kind as would be afforded by observations concerning the fusibility of slags; but perhaps it is still more valuable. As Professor Åkerman justly says, the quantity of heat required to melt a slag (*i.e.*, the total heat of fusion) may be really of more importance to the blast-furnace manager than its actual melting-point. And the total heat of solidification, while probably not exactly proportional to the total heat of fusion, is probably far more nearly so than the melting-point is.

The distinction just made between the total heat of fusion and that of solidification may, perhaps, call for further explanation. These quantities differ for two reasons: *First*, before the silicate can melt, the initially uncombined substances which are to form it must be raised to a temperature at which they

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* *Om Varmbehofven för Olika Masugnsslaggers Smältning*, Stockholm, 1886. As a matter of convenience, I have used the translations, "*Uber die zum Schmelzen verschiedener Hochofenschlacken erforderliche Wärmemenge*," in *Stahl und Eisen*, 1886, vi., p. 281, and the *Journal of the Iron and Steel Institute*, 1886, i., p. 310. Having melted a given silicate, Professor Åkerman allowed it to solidify partly, so as to make sure that the remainder was actually at the freezing point, and then poured part of that which was still molten into a calorimeter, thus obtaining the sum of the latent heat of solidification, plus the heat evolved in cooling from the melting-point down. This sum is the "total heat of solidification."

† Plotting in Fig. 1 the total heat of solidification corresponding to each silicate, I proceeded to draw the contour lines, shown in that figure, in such a way that the silicates through which any given contour-line passed should all have the same total heat of solidification. These contour-lines, therefore, are lines of equal total heat of solidification, and for them I venture to suggest the name "isocal," for lack of a better. "Isotherm" is so closely identified with equal *temperatures* that to apply it to equal *quantities of heat* would be confusing. The measure of temperature is "thermometry," as distinguished from "calorimetry," the quantitative measurement of heat. The only objection to "isocals" is that it is half Greek and half Latin, and therefore to be avoided, unless dictated by advantage or necessity superior to philological considerations. "Calorimetry" itself is open to the same objection; and other instances might be adduced in justification; yet I must confess that I was unwilling to increase the number of such questionable precedents, and that, consequently, I employed, in my first draft of this paper, the term "equical," to take the place of "isocal." But this word, though unobjectionable philologically, was disapproved by friendly critics as sacrificing the self-explanatory analogy (*e.g.*, with isotherms, isobars, etc.) which the prefix "iso" carries with it, in connection with graphic representations of physical conditions. I have, therefore, returned to "isocal," and can only blame the Greeks for having failed to furnish beforehand the means of expressing the difference between degrees of temperature and quantities of heat.

can react on each other and unite; and this, the "formation-point" of the silicate, may be far above its melting-point proper, *i.e.*, the temperature thereafter needed to melt the already formed silicate. *Secondly*, the chemical union of these initially uncombined substances will usually be attended by a considerable evolution of heat. but there will not be, as a general rule, an equivalent absorption of heat on solidification, because these substances do not then dissociate, or, at least, they do not completely return to their original separate condition, and therefore do not reabsorb as much heat as they evolved when uniting.

Details of Construction.—The 65 slags given in Prof. Åkerman's Table III. are represented in Fig. 1. These consist chiefly of silica, lime and alumina. The small quantities of the other substances present were taken into consideration as follows: The magnesia, which varied from 0.62 to 2.01 per cent., was added to the lime, after multiplying it by 1.4, the ratio of their atomic weights. The potash, soda and ferrous oxide, which varied respectively from 0 to 0.96, from 0 to 0.27, and from 0.34 to 0.75 per cent., were deducted from 100, and the lime (plus 1.4 times the magnesia), the silica and the alumina were recalculated as percentages of the remainder. In other words, the magnesia was treated as if it were the equivalent of lime, molecule for molecule, and the alkalies and ferrous oxide were treated as if they were foreign bodies without influence. These assumptions are of course incorrect, and vitiate the results proportionally. But the amounts of these substances were so small that any resulting errors can hardly have affected measurably the small-scale graphic representation given in Fig. 1.

In plotting in Fig. 1 certain slags actually made in running charcoal and coke iron blast-furnaces, the procedure was as follows:

No coke-slags were accepted which contained more than 3 per cent. of manganous oxide. Only the five components, silica, lime, alumina, magnesia and manganous oxide, were taken into account. After multiplying the magnesia by 1.4, the percentage of each of the five components was multiplied by 100 and divided by the sum of the five, so that the sum of the quotients was 100 per cent. The silica and alumina were plotted directly from these quotients. The sum of the lime plus the manganous oxide plus 1.4 times the magnesia was plotted as lime.

General Deductions.—We note in the diagram the narrow wasp-shaped region of greatest fusibility, or, to speak accurately, of lowest total heat of solidification. From this region the isocals rise rapidly in every direction, as we should expect, showing how fast the fusibility diminishes (or, more accurately, the total heat increases) as we depart in any direction from the region of lowest total heat. Characterizing, for convenience, the top of the diagram as "north," we see that, to the northeast and the southeast, they rise much more abruptly than towards the west, southwest and northwest. In other words, between singulo- and subsilicates, even slightly increasing the lime beyond the proportion corresponding to the lowest total heat, and, in case of bi- and tri-silicates, decreasing the alumina below that proportion even slightly, raises the melting-point (more accurately, increases the total heat of solidification) rapidly, far more rapidly, percentage for percentage, than does an increase of silica or alumina, or both.

We next notice that the region of lowest total heat, and also the isocals on its left or west side, lie nearly at right angles with the lime-scale, showing that, in this region (comprised between, say, sub- and tri-silicate, and between lime 30 and 45 per cent.), if the percentage of lime be fixed, substituting silica for alumina or *vice versa* does not materially change the total heat of solidification. Hence, within these limits, the percentage of lime is a far more important thing to bear in mind than that of either silica or alumina, as far as the melting-point of the slag is concerned.

Position of Charcoal- and of Coke-Furnace Slags.—In Fig. 1 there are plotted (1), as small triangles, a number of slags made in iron blast-furnaces using charcoal for fuel, and (2), as small circles, many slags from like furnaces using coke. We at once see that, while the areas covered by these two classes of slags overlap, yet the coke-slags, as a whole, lie much nearer than the charcoal-slags to the lime-corner of the field. Thus, while nearly all the coke-slags lie on the lime-side of the region of minimum total heat (or, to speak loosely, of maximum fusibility)—in some cases very far on that side—the charcoal-slags, in general, lie either only slightly on the lime-side of that region or actually on its silica-side.

The reason for this difference is, of course, familiar to the

iron-metallurgist. In most cases, the gangue is siliceous. Since the slag is composed chiefly of the gangue plus the flux, a siliceous slag is cheaper, in such cases, than a basic one, in that it calls for a smaller addition of basic flux (in practice, limestone). Hence the metallurgist would usually aim at a siliceous slag unless other considerations prevented. But when the fuel is coke, such other considerations do prevent; for the sulphur contained in the coke has to be removed in the blast-furnace, and, in order to remove it, the slag must be very basic, *i.e.*, very rich in lime, or in lime *plus* magnesia. Hence the coke-furnace slags usually lie on the lime-side of the region of greatest fusibility, in spite of the fact that this increases their first cost.

As charcoal is relatively free from sulphur, and as there is therefore usually much less sulphur to be removed in charcoal- than in coke-furnace practice, the slags of charcoal-furnaces usually need not be so basic as those of coke-furnaces must be. Hence, having no sufficient motive for making the more expensive basic slag, the metallurgist may here often use the cheaper siliceous one.

Harmony of Results.—The general parallelism of the isocals gives the best evidence of the trustworthiness of Professor Åkerman's results, and of the great care which he must have taken in this difficult and laborious investigation.

Résumé.—Fig. 1 shows: (1) the general relation between composition and total heat of solidification; (2) that in passing from the region of lowest total heat in the direction of more silica or more alumina, the isocals rise more slowly (*i.e.*, the total heat increases more slowly) than if we pass from the singulo- or sub-silicate region in the direction of more lime; (3) how greatly the presence of a very little alumina lessens the total heat of the bi- and tri-silicates; (4) that between sub- and tri-silicates, and between lime 30 and 45 per cent., silica and alumina can be substituted for each other without materially affecting the total heat of solidification; (5) the relative position of common coke- and charcoal- blast-furnace slags; and (6) the surprising harmony of Professor Åkerman's results. It seems to me that on each of these six points the teaching of the tri-axial diagram is both easier to interpret and more conclusive than that of a di-axial diagram could readily be made.

Triangular Pyramid.—Many years ago, when I began using

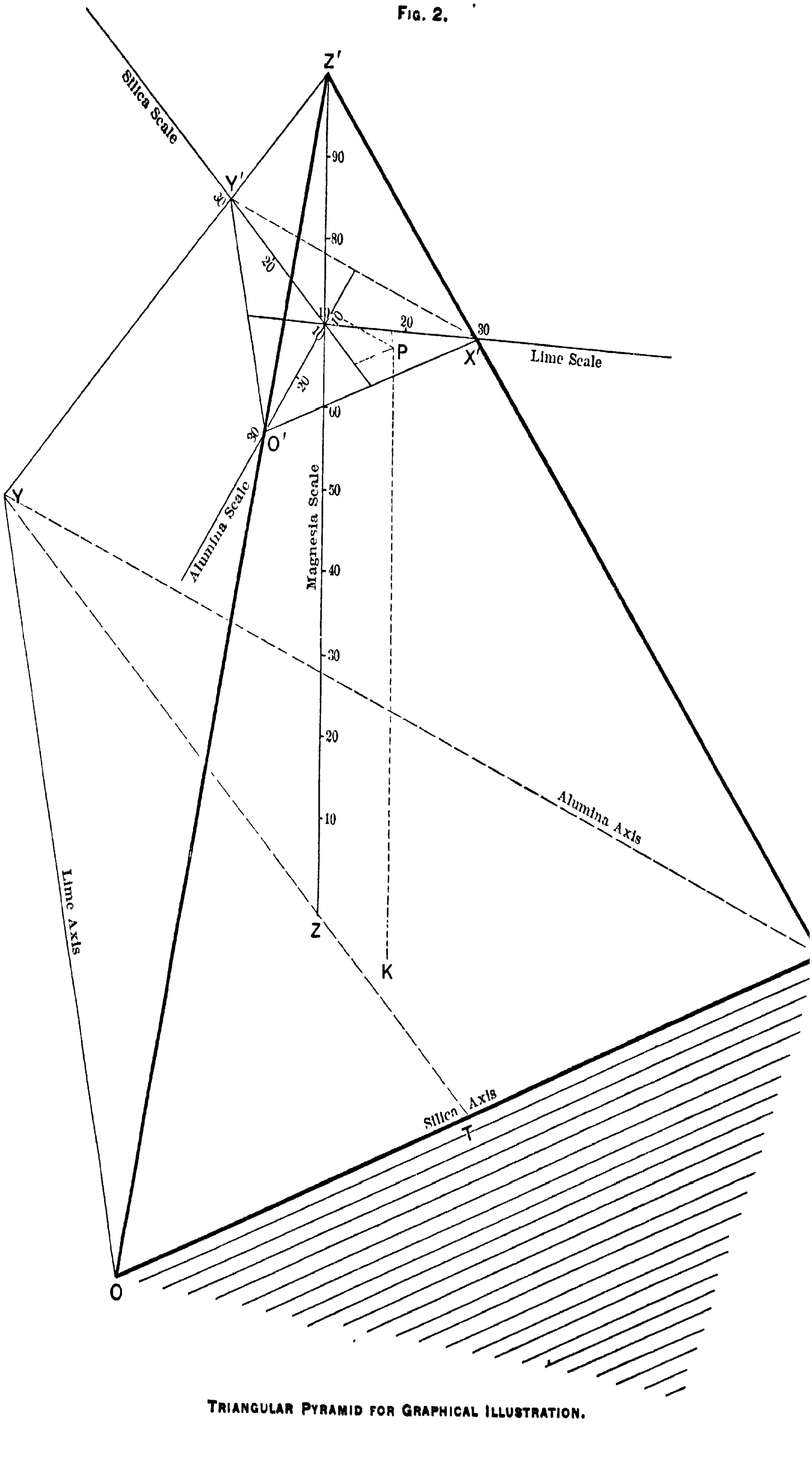
diagrams of this type, it came out in conversation with Professor R. H. Richards that the same principle could probably be extended to the representation of four variables; and, at the suggestion of the Secretary of the Institute, in the colloquial discussion which followed the oral presentation of the main features of this paper at the Atlantic City meeting, I have made a further study of that question.

Bearing in mind that, in any graphic system expressing the variable elements according to percentages, the sum of the co-ordinates representing such percentages must be constant—*i.e.*, always 100 per cent.—and that the tri-axial diagram, based upon the equilateral triangle as a symmetrical system of axes, seems to exhaust the useful possibilities of axes in one plane, we are led to inquire whether a fourth co-ordinate, at right-angles to that plane, could be included in the system. We are thus called to deal with the following problem:

Given a triangular pyramid, with an equilateral base, to find therefor a height such that for all points in the basal plane or within the pyramid, the sum of the four co-ordinates (namely, three perpendiculars drawn from the point, in a plane parallel to the base, to the three sides of the triangle formed by the intersection of that plane with the sides of the pyramid, plus a fourth perpendicular from the point to the basal plane) shall be constant.

Obviously, for points in the basal plane, the co-ordinate perpendicular to that plane would be zero, and the sum of the other three co-ordinates would be, by virtue of the properties of an equilateral triangle, equal to the length of a perpendicular from either apex to the opposite side, or, in other words, to the altitude of the triangle. The problem requires that for any section of the pyramid parallel with the base, the sum of the three similar co-ordinates, plus the perpendicular to the basal plane, shall be the same—that is, shall be the altitude of the basal triangle. Let H be the height of the pyramid sought; x, y and z the co-ordinates of any point in the basal plane; x', y' and z' the similar horizontal co-ordinates of any point above that plane; v the fourth co-ordinate, perpendicular to the basal plane; A the altitude of the triangular base, and A$'$ the altitude of any triangle formed by the intersection of the pyramid by a plane parallel to the base. Then:

FIG. 2.



TRIANGULAR PYRAMID FOR GRAPHICAL ILLUSTRATION.

(1) $x + y + z = A$

(2) $x' + y' + z' = A'$

(3) $A' : A :: H—v : H$

and by the conditions of the problem,

(4) $x' + y' + z' + v = x + y + z = A.$

The value of H is required. From (3),

(5) $A' = \frac{AH—Av}{H}$

Substituting this value in (2) and (4), we have

(6) $\frac{AH—Av}{H} + v = A$, whence

(7) $H = A.$

That is to say, the height of the pyramid which satisfies the condition stated is the altitude of the equilateral triangle constituting its base.

Such a pyramid is represented in Fig. 2. Since the height ZZ′ of this pyramid equals the height TY of its base, one and the same scale can be used for all four variables..

In this figure, the basal plane OXY represents all possible silicates of lime and alumina, and is identical in every respect with the equilateral triangle in Fig. 1. And any plane parallel with OXY represents silicates of lime and alumina, like the basal plane, and also of magnesia, vertical distances representing the percentage of this substance. For instance, the plane O′ X′ Y′ corresponds to 70 per cent. of magnesia, because it cuts the vertical axis ZZ′ of the pyramid at 70 above its base. The divisions of the silica, lime and alumina scales in that and every other plane are of the same length as those of the base triangle OXY, and consequently each of these three scales in the plane O′ X′ Y′ runs only from 0 to 30, instead of from 0 to 100, as those of the base-plane do.

Taking, now, the point P in this plane O′ X′ Y′, it corresponds to the composition:

	Per cent.
Magnesia, ...	70.0
Silica, ...	3.5
Lime, ...	18.0
Alumina, ...	8.5
Total, ...	100.0

Magnesia 70, because the whole plane is 70 above the base; silica 3.5, because the perpendicular distance of P from the

silica axis O′ X′, as measured along the silica scale, is 3.5; and 18 lime and 8.5 alumina for like reasons.

Similarly determined, the four co-ordinates of any point in the pyramid will add up to 100.

Stamp-Mill Indicator Diagrams.

BY PROF. HENRY LOUIS, NEWCASTLE-ON-TYNE, ENGLAND.

(Atlantic City Meeting, February, 1898.)

THE object of the present paper is to call attention briefly to a novel method of analyzing the action of the ordinary gravity stamp, which has not only thrown much light upon the exact motion of the stamp-head, but promises also to be of value in determining the efficiency of a new mill before setting it to its work of crushing, and in this respect should be of service to constructors of mining machinery as well as to mining engineers. The method was devised by Mr. D. B. Morison, and developed in connection with his experiments on the comparative crushing-effects of his patent high-speed stamp and the ordinary cam-stamp, which were carried out at the works of Messrs. L. Richardson & Sons, Engineers, Hartlepool, and on which he read a paper at Newcastle in 1897.* Mr. Morison's experiments were of a very exhaustive nature; and having been consulted by him during his investigations, I had frequent opportunities of examining the methods employed and the results obtained.

Experiments were first made with an apparatus resembling the indicator of a steam-engine, inasmuch as the drum revolved alternately backwards and forwards, thus producing a closed diagram. This form was not found satisfactory, and it was therefore altered so as to cause the drum to revolve continuously in one direction, thus giving a continuous or open diagram. The arrangement is shown in plan in Fig. 1; a collar, A, surrounds the stamp-stem loosely and works between the guides B, B, which constrain it to move in a vertical line only. Immediately above it and below it, a couple of collars are clamped to the stamp-stem, so that the latter, while free to revolve,

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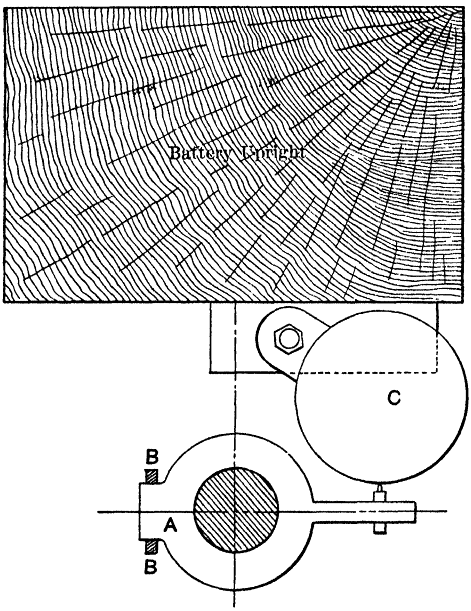
* *Proceedings of the North-East Coast Institution of Engineers and Shipbuilders*, vol. xiii.

communicates its vertical motion to the piece A. From A an arm projects, carrying a pencil that presses against the drum C, 7 inches in diameter, driven at a uniform rate by a cord off the cam-shaft. The relation between the vertical movement of the stamp and the rotation of the cam-shaft was thus obtained graphically, while the latter uniform motion was readily expressed in terms of time by noting the rate of revolution, which was carefully kept unaltered during each test, the speed being the average recorded by three independent observers.

FIG. 1.



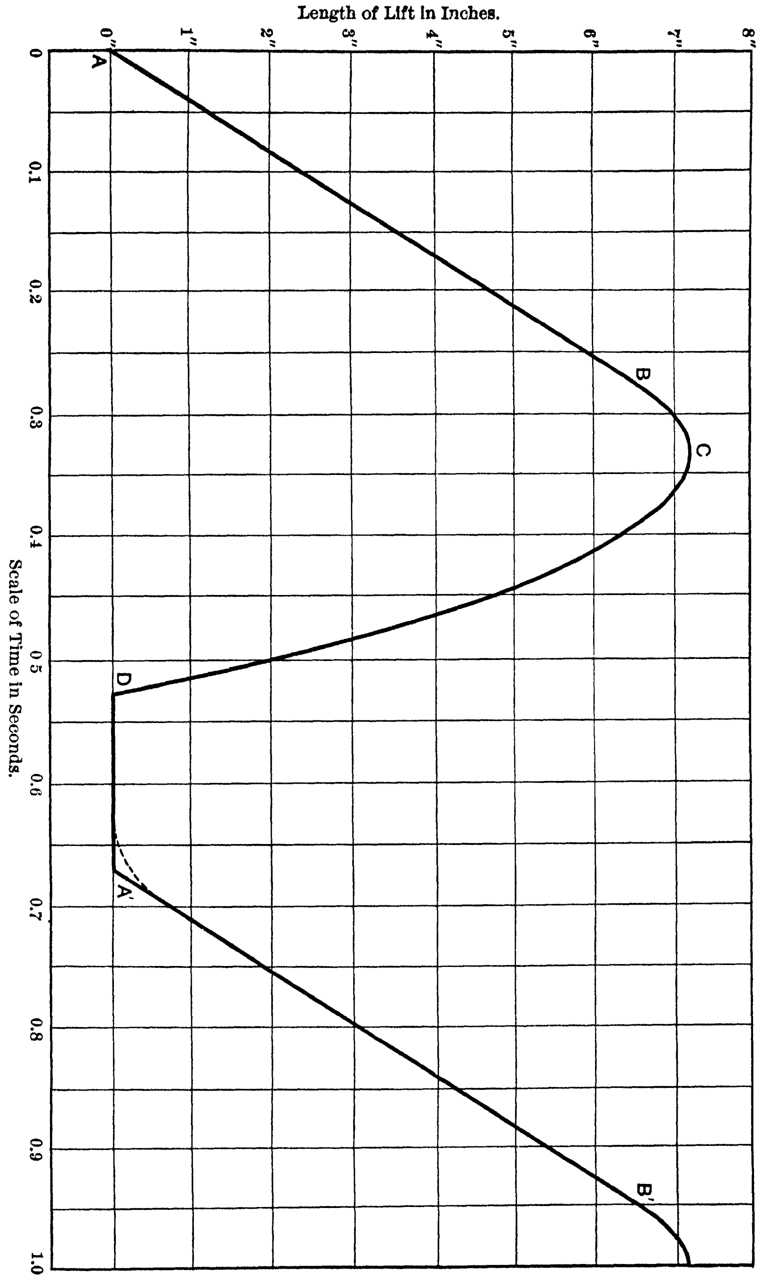
PLAN OF STAMP-STEM AND INDICATOR.
Scale 1½″ to 1 foot.

The degree of accuracy thus attainable with ordinary care is quite sufficient for all practical purposes.

The stamp employed was the ordinary pattern, manufactured by the Sandycroft Foundry Company, Hawarden, England, its chief dimensions being as follows: Diameter of stem, 3¼ inches; length, 13 feet; total weight of stamp, 900 pounds; distance from center of cam-shaft to center of stamp-stem, 5 inches; diameter of cam (toe to toe), 20 inches; depth of top and bottom guides, 14 inches each.

The nature of the results obtained is shown by means of the

Fig. 2.



IDEAL STAMP-MILL DIAGRAM. 7 IN. LIFT; 90 DROPS PER MINUTE.

subjoined diagrams, Figs. 3 to 8, which are a few of the extensive series taken by Mr. Morison. They may be divided into two sets, the first taken when the stamps were set with a dead

lift of 6.5 inches, the second with a dead lift of 8 inches, and their speeds were as follows:

	Lift. Inches.	Speed. Drops per minute.
Fig. 3, ...	6.5	82
" 4, ...	"	88
" 5, ...	"	97
" 6, ...	8	80
" 7, ...	"	84
" 8, ...	"	93

In order to explain more readily the meaning of these diagrams, I have drawn in Fig. 2 an ideal stamp-mill diagram, or the diagram that would be obtained from a stamp-mill working in vacuo absolutely without friction, either of the stem against the guides, etc., or of the shoe against the pulp in the battery-box, and assuming that the whole of the momentum of the crushing-head is instantaneously converted into crushing-effect, so that it strikes quite dead and produces no rebound of the stamp at all—conditions which, it need hardly be said, can never obtain in practice. I have assumed that the distance from the center of the cam-shaft to that of the stamp-stem (the radius of the generating circle of the cam) is 5 inches, that the lift of the cam is 7 inches, and that the stamp is making 90 drops per minute.

The time of one complete lift + drop is therefore $60 \div 90 = 0.667$ second. The time of the lift* is $\frac{7}{5\pi} \times \frac{60}{90} = 0.297$ second.

As the height of the lift is 7 inches, the speed of the lift is at the rate of 1.96 feet per second. Starting, therefore, at the moment when the cam is just about to engage with the tappet at A on the diagram, the uniform velocity produced by a properly designed cam is shown by the straight line A B. It is usual with most makers of stamp-mills to flatten the curve of the toe of the cam so that the stamp shall not leave the cam at too high a speed, and this flattening should extend to the last half-inch of the lift of the cam, and should be such as to correspond to the velocity with which the stamp would con-

* For the formulas used in these calculations (which are, however, fairly well known), I must refer to my *Handbook of Gold-Milling* (Macmillan & Co.), 1894, page 190.

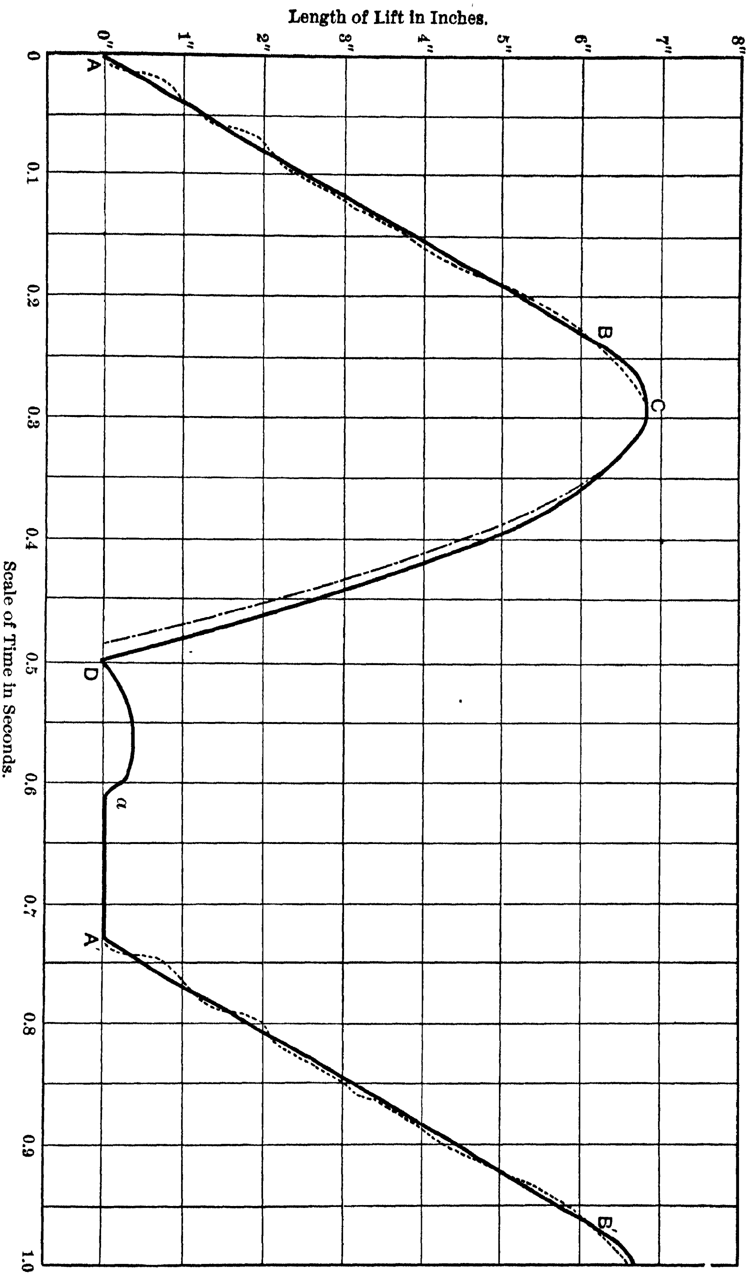
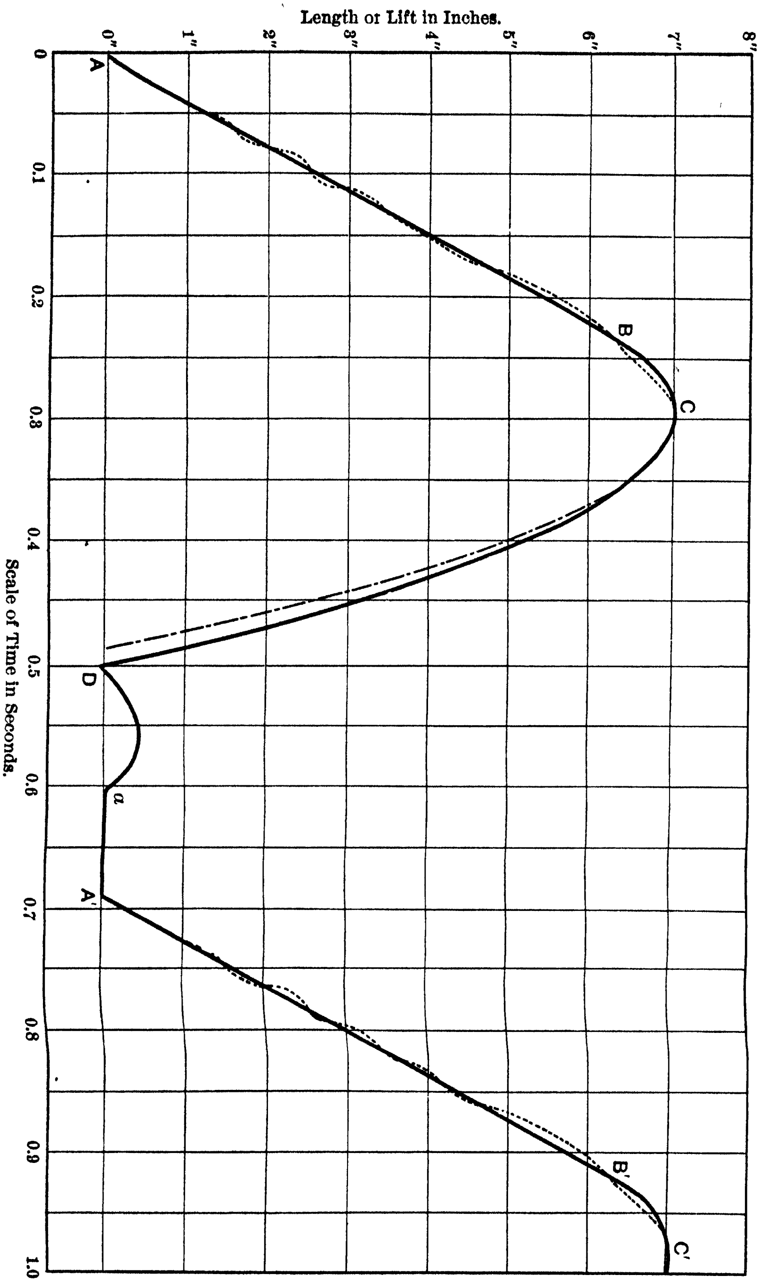


FIG. 3.

STAMP-MILL DIAGRAM. 6½ IN. LIFT; 82 DROPS PER MINUTE.

tinue to move if under the influence only of the velocity imparted to it by the cam and of gravity. In all ordinary mills friction comes into play, but in our ideal cam-curve this force

FIG. 4.



STAMP-MILL DIAGRAM. 6½ IN. LIFT; 88 DROPS PER MINUTE.

is to be neglected. Hence, after the stamp has risen from A to B, or has completed 6.5 inches of its lift, its uniform rate of motion becomes modified by the action of gravity, just as

though the cam had lifted it through 6.5 inches only instead of 7 inches. At B the stamp was moving at the rate of 1.96 feet per second, and will therefore continue to rise for $\frac{1.96}{32.2} = 0.0608$ second, during which it will rise to a height of 0.0595 foot or 0.714 inch. From these data it is easy to set out C, or the point where the stamp comes to rest and commences to fall, and C D can then be drawn as an ordinary gravity-curve corresponding to a period of 0.194 second, D being the point of time at which the stamp strikes the layer of stone upon the die; it then continues at rest until 0.667 second from the commencement of its lift has elapsed, or for a time equal to 0.667 — (0.276 + 0.061 + 0.194) = 0.136 second. At the end of this period, at A′, the cam once more touches the tappet and a second lift commences. Some makers alter the shape of the base of the cam-curve, so as to make the cam engage the tappet more gradually, in which case the dotted line shown will more correctly represent the diagram of the lift. This precaution is advisable more especially when the stamp is to work at a high rate of speed.

Turning now to the actual diagrams taken, it is interesting to note the differences between them and the ideal diagram. In Fig. 3, the line of lift commences at A (the same letters are used throughout to represent similar parts of each diagram) and continues to B, giving apparently an actual lift at uniform speed to a height of 6⅛ inches. Of course, in the real stamp, the toe of the cam is doing work, because, when correctly designed, it is engaged in overcoming the friction of the stem against the guides, etc.; it should not force the stamp up any more rapidly than it would move if under the influence of gravity alone, as in Fig. 2; nor, on the other hand, should it allow the effect of friction to diminish its velocity to any greater extent than is there indicated. It will be noticed that the line A B is not exactly straight, but shows a certain amount of waviness. This is due, either to small imperfections in the shape of the cam, or to vibration of the mill-framing. The dotted line in Fig. 3 and the subsequent figures shows the actual diagram obtained, the full line being obtained by "smoothing" the undulations. The stamp continues to rise by reason of its acquired velocity up to C, a height of 6.85 inches, and then falls again. The theoretical gravity curve of a body fall-

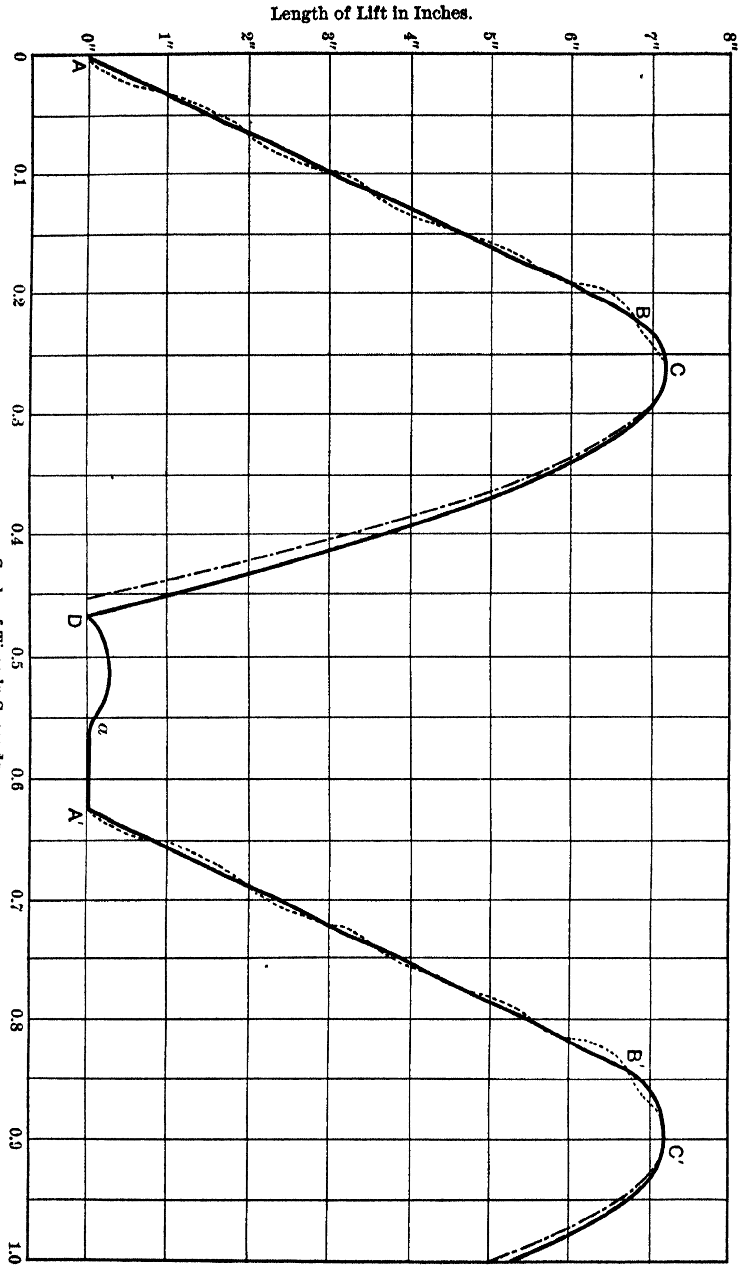
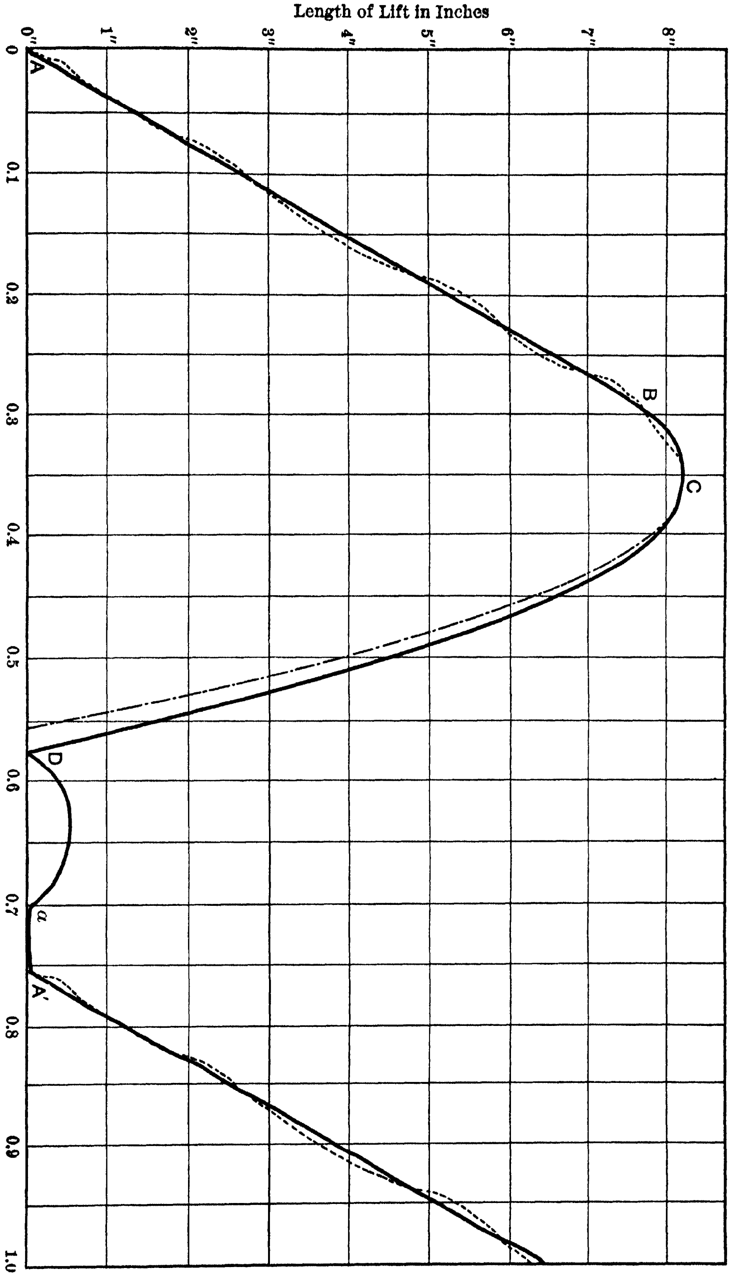


FIG. 5.

STAMP-MILL DIAGRAM. 6⅛ IN. LIFT; 97 DROPS PER MINUTE.

ing without any retardation from friction has been plotted on the same diagram for the purpose of comparison, and it will be seen that the stamp falls more slowly by an amount equal to

Fig. 6.



STAMP-MILL DIAGRAM. 8 IN. LIFT; 80 DROPS PER MINUTE.

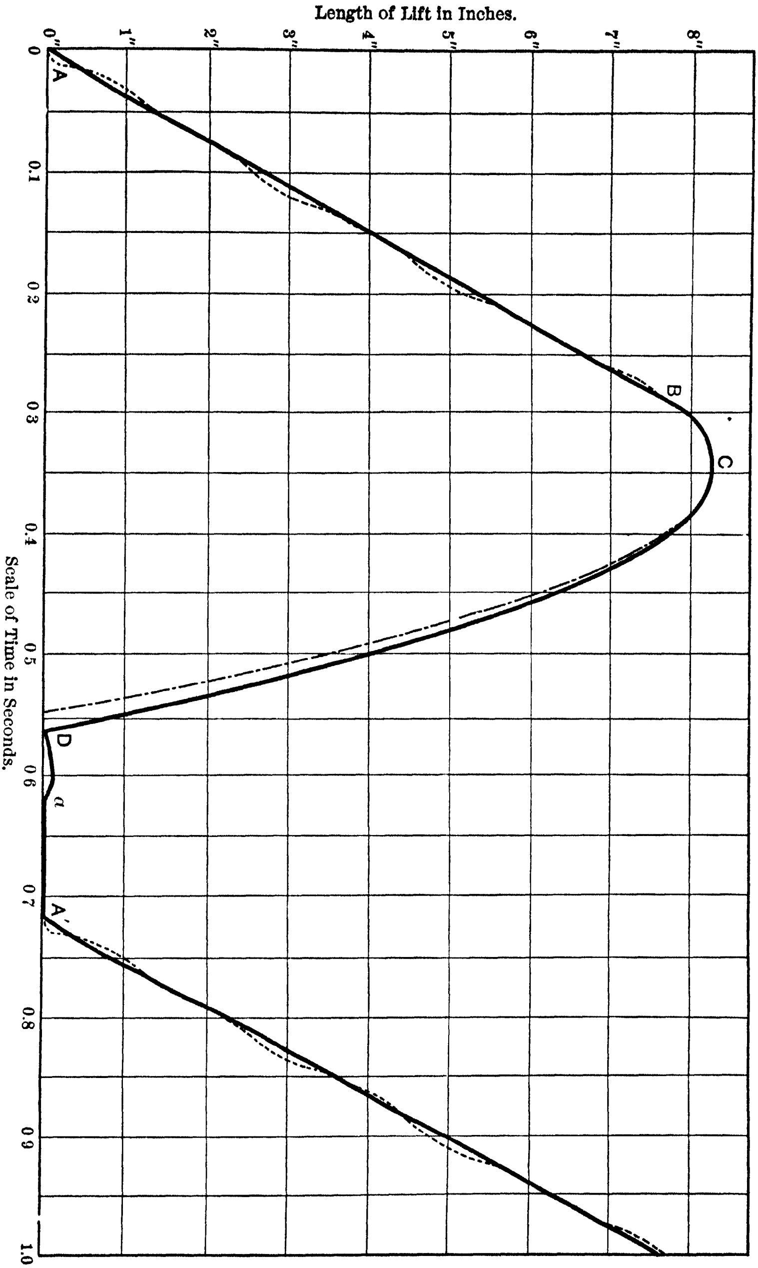
about 0.015 second. The theoretical time of falling from a height of 6.85 inches would be 0.19 second, and the velocity at the end of that time would be 6.1 feet per second. The ac-

tual velocity, as shown in the diagram, is only about 5.2 feet per second. As it is the momentum of the stamp that determines its crushing-capacity, this determination of its final velocity is a matter of some importance, and as the velocity here obtained would be attained by a frictionless body in falling from a height of 5.05 inches only, the amount of power wasted by friction of all kinds amounts in this particular case to 26 per cent. of that exerted in lifting the stamp. An examination of the curves will show the reason why the loss of power is so great. At the commencement of the fall the actual diagram-curve and the theoretical gravity-curve practically coincide, the difference between them being more marked as the fall proceeds and as the velocity of the stamp increases. This result is in accordance with the well-known law that the retarding effect of friction increases more rapidly than the velocity, or approximately as the square of the velocity.

On reaching D it will be noticed that the stamp descends slightly, but only very slightly, below the zero-line, owing to the compression of the material lying on the dies, or of that of the stamp itself, or of the die, mortar-box and foundations. Immediately recovering, it at once rebounds at a comparatively slow speed to a height of 0.36 inch, and then falls again to its normal position at *a*, continuing at rest until a fresh lift begins once again at A′. The distance from D to A′ corresponds with the interval of rest of the normal diagram, and amounts in this case to 0.23 second. As the time of one complete lift is $60 \div 82 = 0.731$ second, the interval of rest amounts to 31.4 per cent. of the total period of working.

Turning next to Figs. 4 and 5, we note that the phenomena shown by the diagram are the same in character, and differ only in degree. As was to be expected, the stamp, moving with the greater velocity due to the increased number of drops, is thrown up higher before it comes to rest, and the irregularities in its lift are slightly more marked. The ultimate divergence between the diagram of the actual fall and the theoretical gravity-curve is also progressively greater, owing to the increase, in each case, of the distance fallen through. The amount and the rate of rebound seem to vary independently of the speed of the cam, and this was only to be expected, seeing that the velocity of the impact of the stamp is only one of many factors af-

FIG. 7.



STAMP-MILL DIAGRAM. 8 IN. LIFT; 84 DROPS PER MINUTE.

fecting the rebound, and by no means one of the most important. The total interval of rest grows less as the speed increases, as was again to be expected.

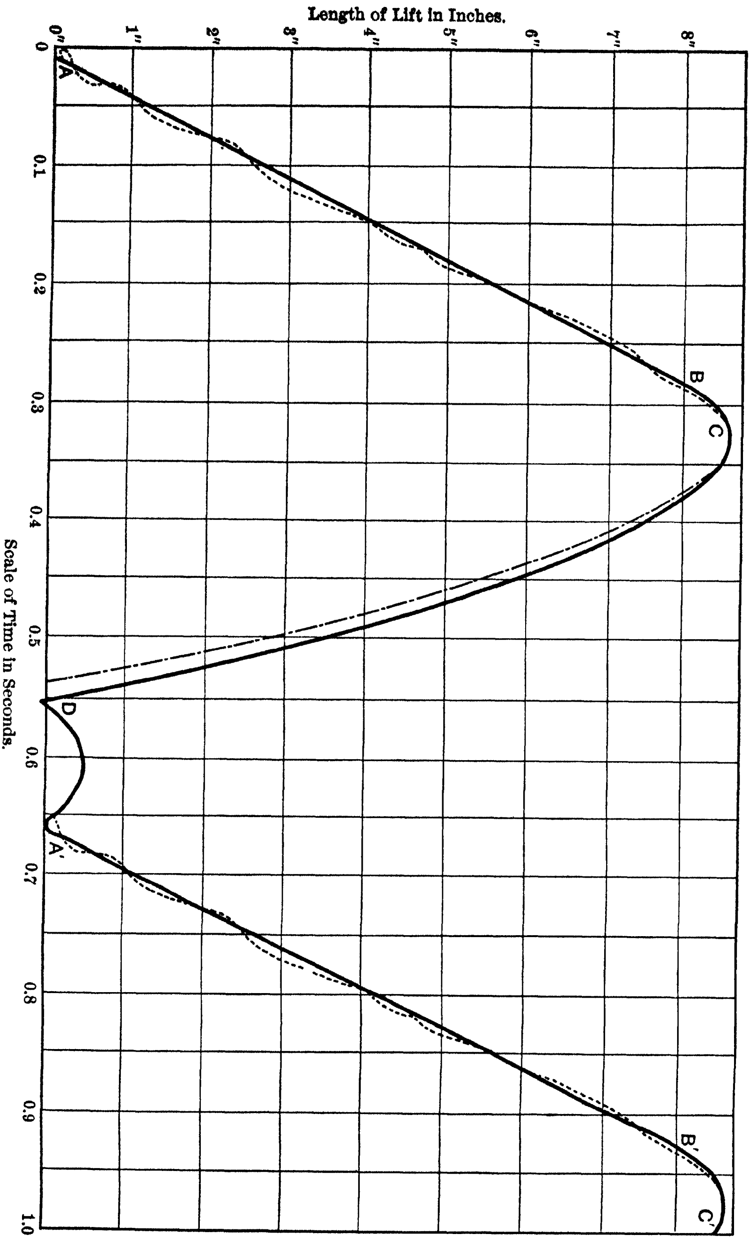


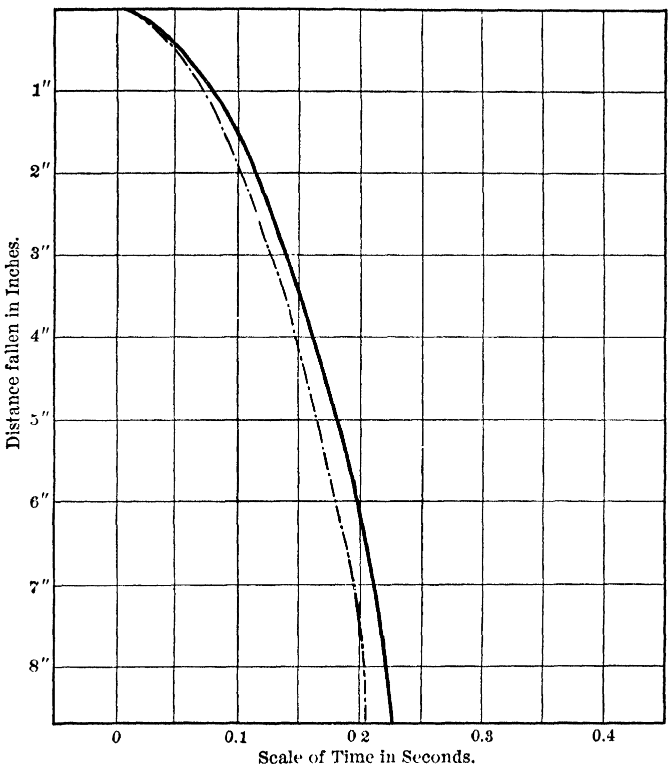
FIG. 8.

STAMP-MILL DIAGRAM. 8 IN. LIFT; 93 DROPS PER MINUTE.

An examination of Figs. 6, 7 and 8 shows exactly the same series of results, only exaggerated, because, the length of drop being greater, there is more time given for the development of the various forces that affect the result. In Fig. 8 it will be

noticed that the lift actually commences before the stamp has quite come to rest after its rebound. In other words, the speed of the stamp has exceeded its maximum permissible limit; slight "camming" is taking place, and the conditions are such that, if this speed were continued, something would break, most probably one of the arms of the cam. It may be noticed that the interval of rest here amounts to almost exactly 0.1 sec-

FIG. 9.



TIME OCCUPIED BY DROP OF NORMAL STAMP, COMPARED WITH RATE OF FALLING IN VACUO.

ond. In the book I have previously referred to I wrote (page 191): "The minimum interval of rest which can be allowed in a stamp-mill is one-tenth of a second." This statement was the average of a number of rough tests made on stamp-mills in actual running, and it is satisfactory to find my rough approximation so closely corroborated by the data furnished by the more scientific method of the indicator-diagram.

The following table shows the more important of the numerical results derived from the indicator-diagrams, assembled for convenience of reference. The first two columns are given by the conditions of the experiment. Column III. is calculated from No. II. Column IV. is obtained by direct measurement. It must be borne in mind that the figure obtained from the theoretical diagram (Fig. 2) in this column cannot well be compared with the others, because it is derived from an imaginary cam-curve, which shall not affect the tappet after the latter has been raised 6.5 inches, whereas the other numbers are obtained by actual experiment with a cam that has not been designed to fulfill exactly this condition. Moreover, in the ideal diagram, the effect of the friction between the cam and tappet has purposely been neglected. Column V. is obtained by direct measurement from the diagram. Column VI. gives the ratio of the actual velocity of impact to the theoretical velocity that would be obtained if the fall of the stamp were not retarded by friction. The square of this fraction gives the proportion of the force employed in lifting the stamp, the remainder being wasted by friction in falling. The whole of this loss can by no means be set down to imperfection of the machinery. In addition to friction against the guides, which can be reduced to a very small figure, it also represents the friction of the stamp against the air and against the pulp in the battery-box. I am inclined to think that the latter item is by far the most important:

TABLE I.—*Numerical Data, from Figs. 2 to 8 Inclusive.*

Number of Diagram.	I. Height of Lift.	II. Number of Blows per Minute.	III Time of a Complete Blow (Lift and Drop).	IV. Total Height of Fall.	V Interval of Rest	VI Actual Ultimate Velocity ÷ Theoretical Ultimate Velocity.
	Inches.		Second.	Inches.	Second.	
Fig. 2	7	90	0.667	7.21	0.136	1.
3	6½	82	0.731	6.85	0.230	0.85
4	6½	88	0.682	7.05	0.185	0.82
5	6½	97	0.618	7.18	0.159	0.77
6	8	80	0.75	8.23	0.177	0.67
7	8	84	0.714	8.35	0.160	0.66
8	8	93	0.645	8.60	0.100	0.63

Another matter of interest that comes out from these dia-

grams is the actual rate at which a stamp drops in practice. According to the formula for the rate of falling of a body *in vacuo*, if H is the height from which it falls in feet, and t the time in seconds:

$t = \sqrt{\frac{2\,H}{g}}$, where g is the accelerating action of gravity, equal to about 32.2 feet per second.

From a comparison of a number of stamp-diagrams I find that within the ordinary limits of stamp-mill practice, say up to a drop of 9 inches, the actual time T occupied in falling may be very accurately found from the formula:

$T = \sqrt{\frac{2\,H}{K}}$, where K is a coefficient, the mean value of which I find to be 27.5, so that the above formula may be written:

$$T = \frac{\sqrt{H}}{3.71}.$$

From these formulas the following table has been compiled, showing the average rate of falling of an actual stamp, the theoretical rate of a body falling *in vacuo* being added for comparison:

TABLE II.—*Actual and Theoretical Rates of Fall.*

Length of Drop.	Normal Time Occupied by Drop of Stamp.	Theoretical Time of Fall *in vacuo*.
Inches.	Second.	Second.
1	0.078	0.072
2	0.110	0.102
3	0.134	0.125
4	0.154	0.143
5	0.174	0.161
6	0.190	0.176
7	0.206	0.190
8	0.220	0.204
9	0.233	0.216

The curves thus obtained are plotted in Fig. 9 on the same scales as the previous diagrams, the full line showing the actual rate, and the dotted line the theoretical rate of falling. A comparison of the former line with those taken actually from the stamp will, I think, show a sufficiently close approximation to warrant the adoption of my formula in practice. I venture to hope that this brief paper may be the means of inducing others who are in a position to do so to take diagrams from stamp-

mills under all conditions of running, and I shall be grateful for copies of diagrams so taken, to enable me to determine whether the formula given above is universally correct.

Note on the Forms Assumed by the Charge in the Blast-Furnace, as Affected by Various Methods of Filling.

BY FRANK FIRMSTONE, EASTON, PA.

(Buffalo Meeting, October, 1898.)

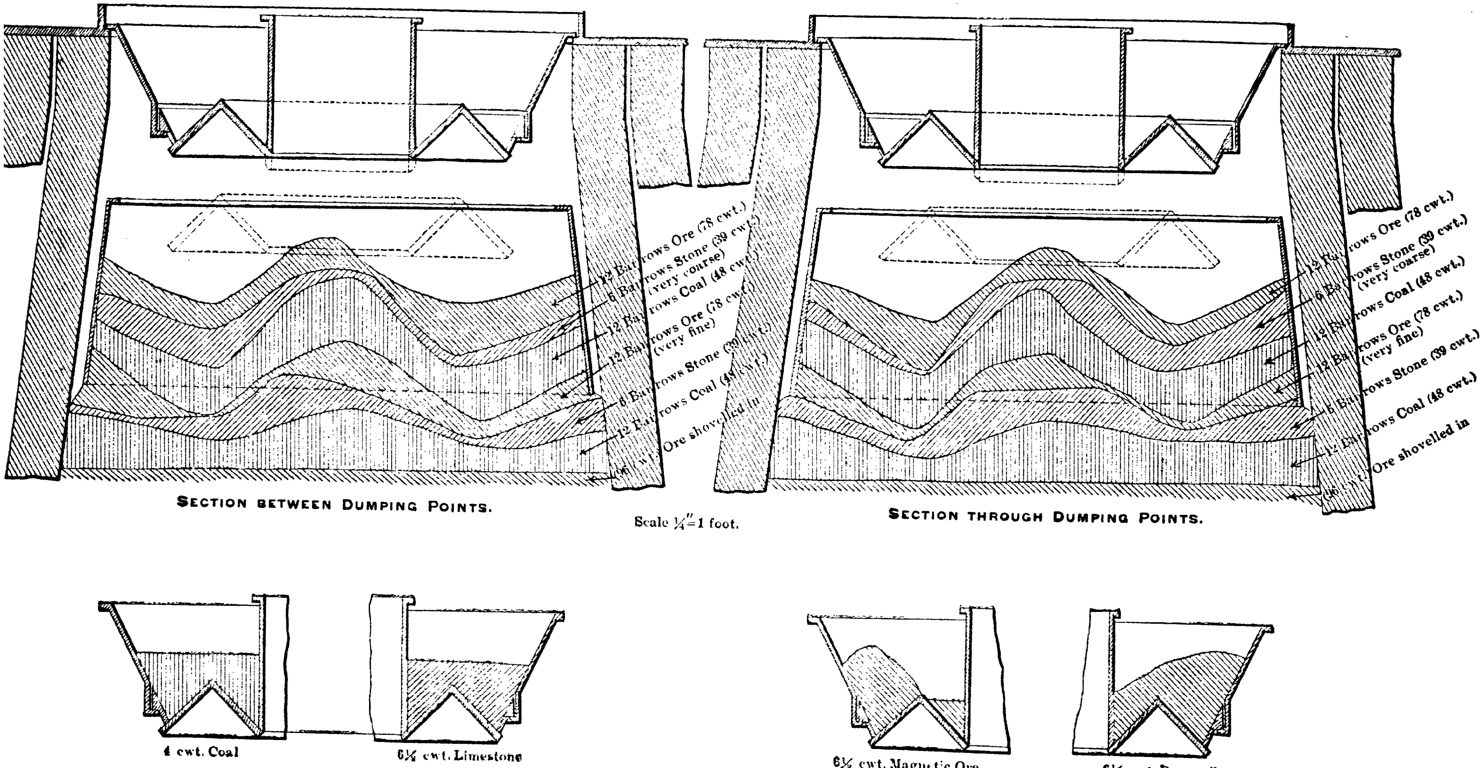
WHEN in charge of the Glendon Iron Works, the importance of good methods of filling was forcibly brought to my attention, and it occurred to me that the first step toward the discovery of the best plans was the accumulation of exact information as to the distribution actually effected by various chargers; also, that something could be done in this direction by careful measurements in the furnace, to be afterwards recorded for comparison on drawings.

The figures shown herewith have been prepared from such measurements, and nearly all of them refer to Glendon, where the fuel used was anthracite and the ore-mixture about three-quarters of New Jersey magnetic ores and one-quarter of brown-ores from the vicinity of the works. Every occasion furnished by the blowing-in of furnaces was utilized, but the data accumulated rather slowly in this way; and, opportunity offering to make special trials at no great expense, the figures from 2 to 7 were thus obtained.

The trials were made by dumping a hopperful (6 barrows) of material at once into the furnace, and, when all the fuel of a round (12 barrows) had been put in, the profile of the upper surface was obtained by stretching a steel tape across the top on a diameter and at a known height. By measuring down from this with a graduated rod, the profile was easily obtained with sufficient accuracy, especial pains being taken to get correctly the crests of the ridges and the bottoms of the hollows, as many intermediate points being included as seemed necessary.

This was repeated on another diameter when there was occasion. The same course was followed for the limestone and ore.

Fig. 1.



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The drawings do not, and in fact cannot, give any information as to how much of each kind of material passes into the interstices of the layer already in the furnace, or as to the actual quantities of fine and coarse stuff in each layer; and the disturbance to the layer already in the furnace when the pieces in the succeeding hopperful fall on it is also of necessity neglected. This I am sure is quite insensible as regards the displacement of coal by limestone or of the latter by the succeeding charge of ore, and I do not think it is great even when coal falls on ore; in the figures from the coke-furnace measured I am certain that it is absolutely negligible.

It is very desirable to know exactly the amount going into the interstices, but I have never been able to devise a method of determining it; and the same is true of a quantitative representation of the distribution of the fine and coarse pieces in the same layer, although it is easy to see differences qualitatively and to note them on the drawing; moreover, in all cases the crests of the ridges are formed of the finer particles, the coarser rolling into and occupying the bottoms of the hollows.

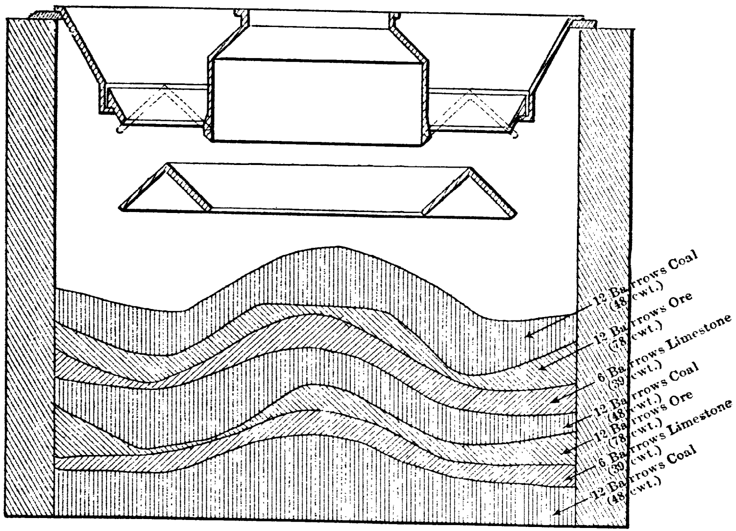
Fig. 1 was measured in No. 5 furnace at Glendon, the bell having the same size as in Fig. 2.

Sections are given on two diameters, one passing through two opposite ones of the six dumping-points and the other exactly between them. There is no visible constant difference in the profiles, and, in fact, there is no appreciable sidewise rolling in hoppers of this size, nor in those of 11 feet diameter in Figs. 3 to 5, for which reason only one section of each is shown, although two were measured and drawn in most cases.

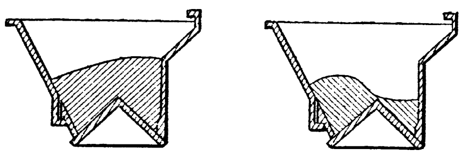
This charger (for a furnace 10 feet in diameter on top) gave excellent results at Glendon on furnaces varying from 15 to 18 feet greatest diameter, and like good results at Stanhope, New Jersey (also with anthracite), and is now in use on the coke-furnaces at Longdale, Virginia; but as the results at Glendon were particularly good with a furnace of 16 feet greatest diameter, it seemed rational to increase the size of the top for the furnaces 18 feet in diameter, and for them the top was accordingly made 11 feet in diameter, and the bells received the proportions shown in Figs. 3, 4 and 5.

It should be observed that, with these "double-bell" chargers,

Fig. 3.



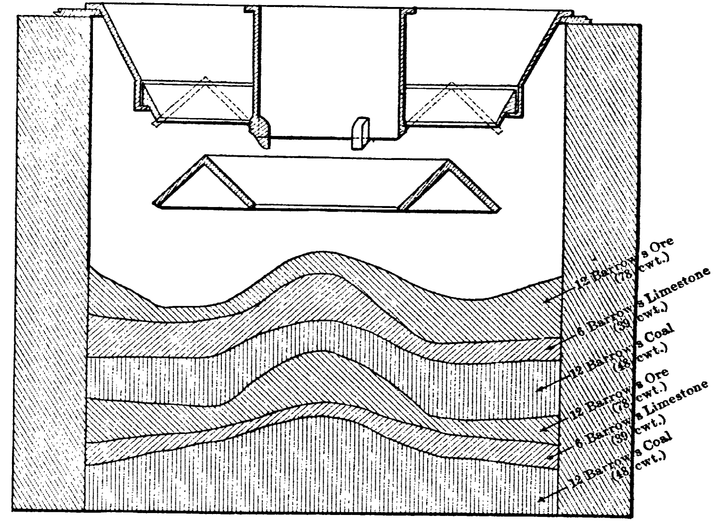
Cross-Section over Barrows.



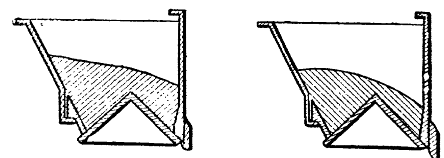
6½ cwt. Stone

6½ cwt. Ore

Fig. 2.



Cross-Section over Barrows.



6½ cwt. Stone.

6½ cwt. Magn. Ore.

and also with the modified Langen, the fraction of the material, especially of the ore and limestone, which passes through the central opening, depends, to some extent, on the way in which it is deposited in the hopper by the barrows (for which reason, sections of the hopper before dumping are shown on the figures), and also that this fraction may be decreased by increasing the amount of lost motion in the links suspending the central plug, thereby decreasing the area of the central opening and delaying the beginning of the central discharge; but with the dimensions on the drawings this can be done only to a limited extent (at least in the case of anthracite), on account of the increased risk of large pieces of fuel jamming between the center-plug and the bell, and preventing the prompt closing of the latter.

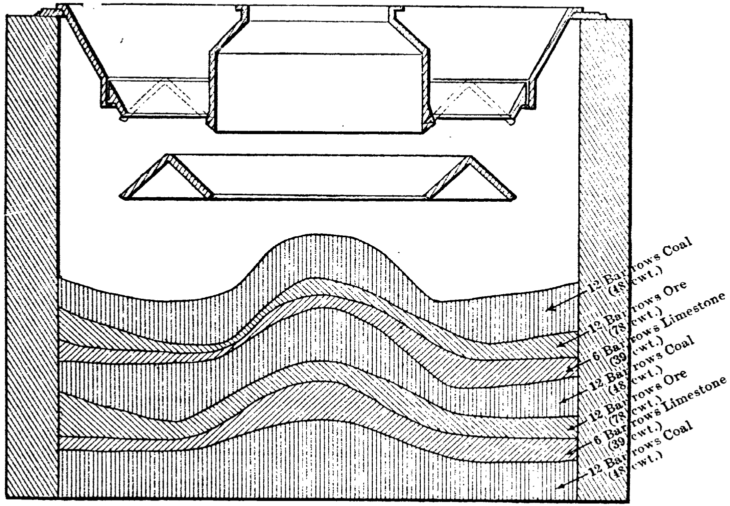
In Fig. 3 the barrow-wheels were allowed to touch the hopper, but in Fig. 5 they were kept back 5 inches from it, which more nearly corresponded to the actual conditions in the furnace.

The furnace-results were good, but not noticeably better than in 18-foot furnaces having 10-foot tops, and the double-bell chargers were soon after superseded by the modified Langen charger which I have described in *Trans.*, xiii., 520.

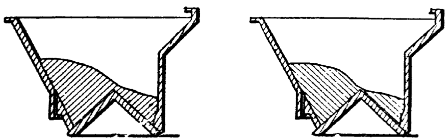
Fig. 6 was made with the bell first used at Glendon, which gave very bad results (*Trans.*, iv., 128). The most important points to notice are the position of the coarse materials next the walls and the complete absence of limestone in the center. The irregular distribution of coal and ore shown on the profile was caused by the bell swinging from side to side instead of descending vertically. This must clearly introduce great irregularities, which may show plainly in the furnace-work, especially when, for any reason, the bell tends to swing more frequently in one direction than in others. This did in fact happen at one of the Glendon furnaces, where, for accidental reasons, the bell always tended to swing in one direction, and, in consequence, the cinder was alternately black and gray in successive flushes, with corresponding irregularity in the iron, until the trouble was detected and remedied.

Fig. 7 was obtained by converting the double bell of Fig. 2 into a single bell, by covering the central opening with a boiler-plate cap; as in Fig. 6, there is no limestone at the center, but the fine stuff is next the walls.

Fig. 5.

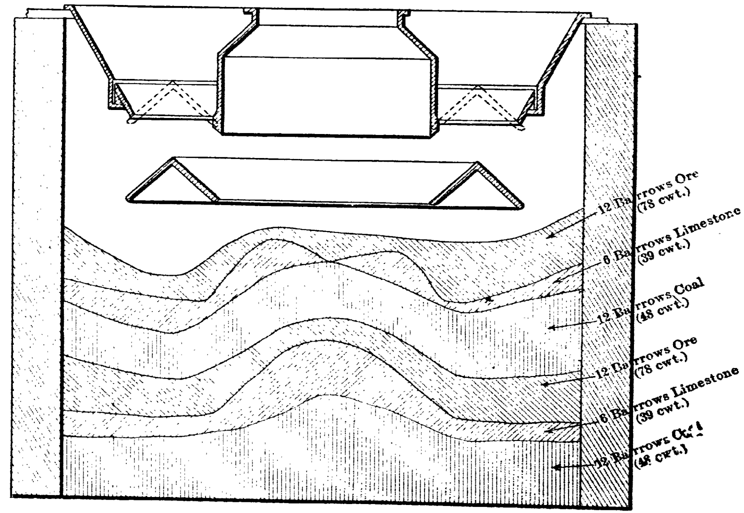


Cross-Section over Barrows.



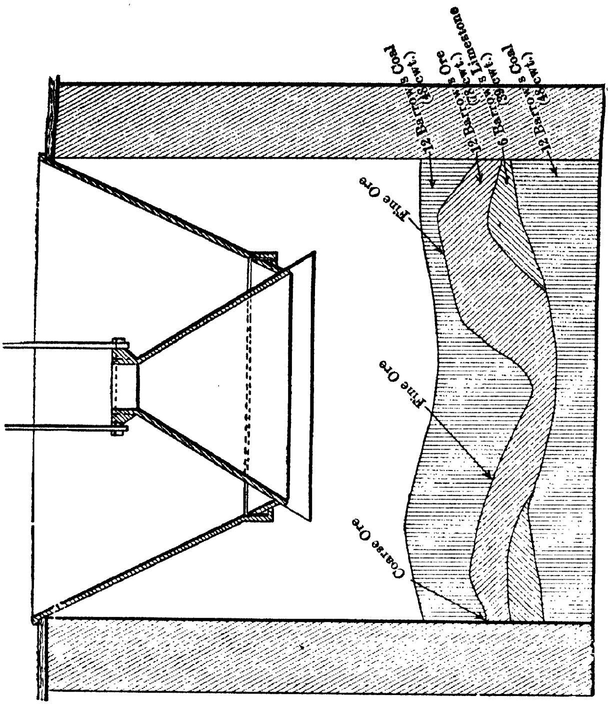
Barrows 5'' Back

Fig. 4.



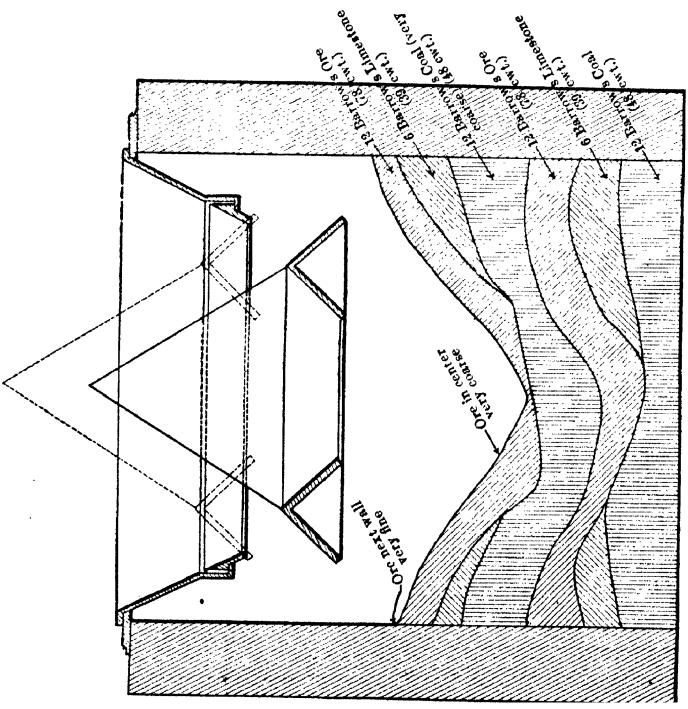
Cross-Section over Barrows.

FIG. 8.



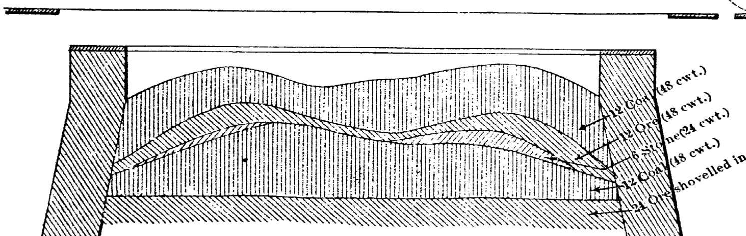
CROSS-SECTION OVER BARROWS.

FIG. 7.

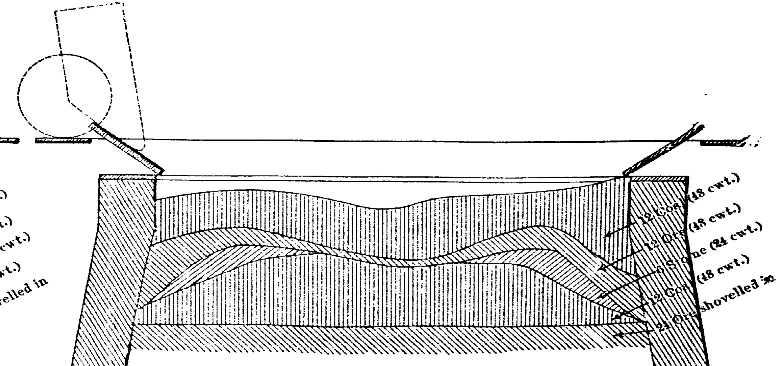


CROSS-SECTION BETWEEN BARROWS.

Fig. 8.



Section between Doors



Section through Doors

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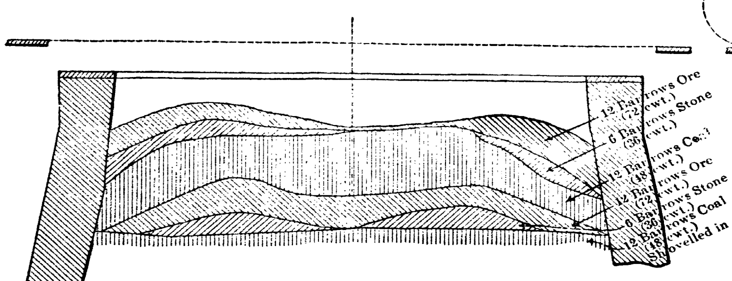
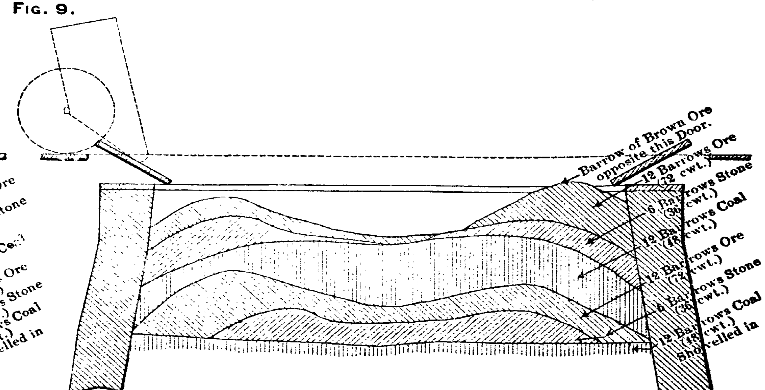
Fig. 9.



Section between Dumping Points.

Scale ¼″=1 foot.



Section through Dumping Points.

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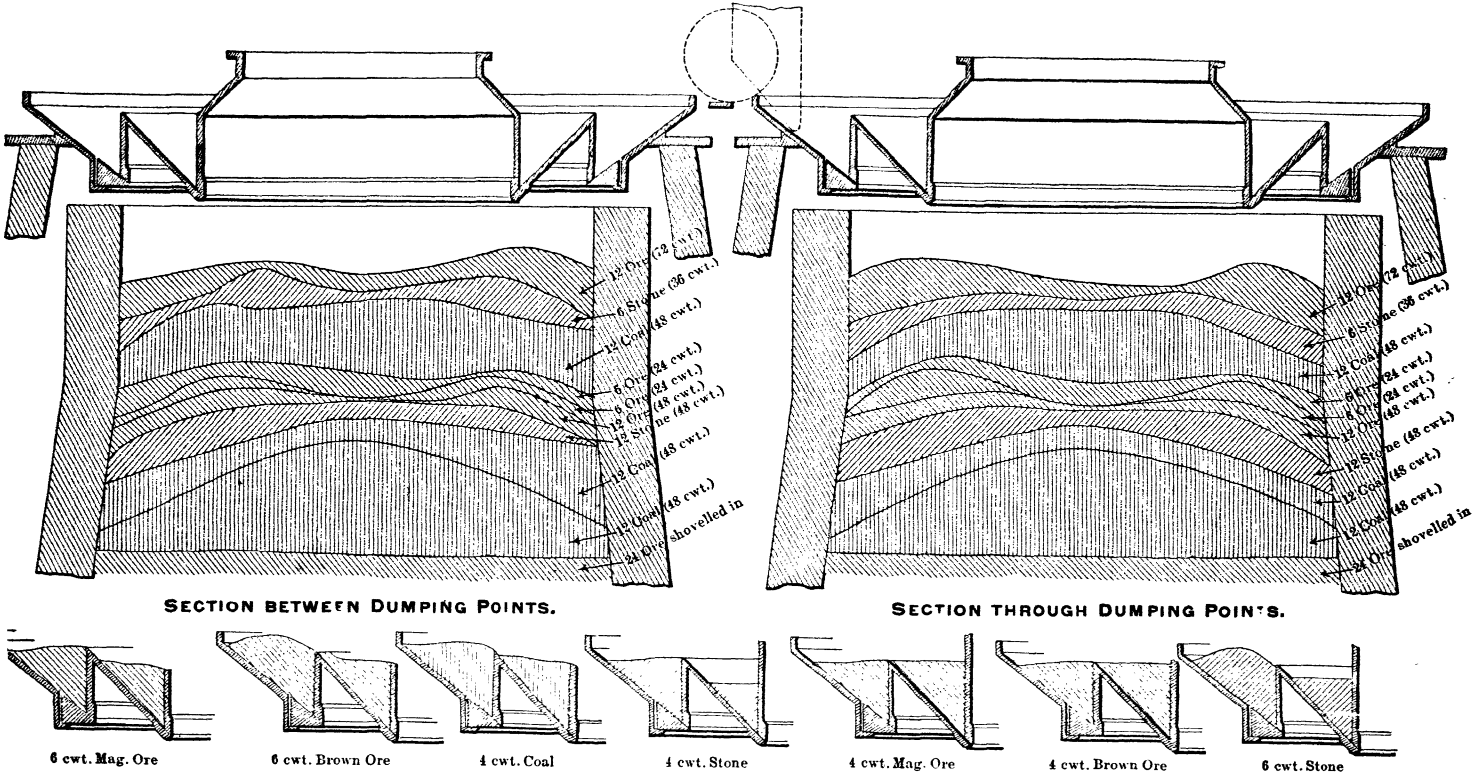
Fig. 10.


12 Ore (72 cwt.)
6 Stone (36 cwt.)
12 Coal (48 cwt.)
6 Ore (24 cwt.)
6 Ore (24 cwt.)
12 Ore (48 cwt.)
12 Stone (48 cwt.)
12 Coal (48 cwt.)
12 Coal (48 cwt.)
24 Ore shovelled in
SECTION BETWEEN DUMPING POINTS.
12 Ore (72 cwt.)
6 Stone (36 cwt.)
12 Coal (48 cwt.)
6 Ore (24 cwt.)
6 Ore (24 cwt.)
12 Ore (48 cwt.)
12 Stone (48 cwt.)
12 Coal (48 cwt.)
12 Coal (48 cwt.)
24 Ore shovelled in
SECTION THROUGH DUMPING POINTS.
6 cwt. Mag. Ore
6 cwt. Brown Ore
4 cwt. Coal
4 cwt. Stone
4 cwt. Mag. Ore
4 cwt. Brown Ore
6 cwt. Stone

I have no experience as to the working of a furnace with such a bell; no doubt it would be much better than with Fig. 6, but No. 2 furnace at Stanhope, with a bell much larger than Fig. 6, although not so large as Fig. 7, was greatly improved by substituting the double bell of Figs. 1 and 2.

Figs. 8 and 9 were obtained from No. 4 furnace at Glendon. This furnace, standing by itself and blown by water-power, was never altered to the closed-top plan, and therefore furnished a standard of comparison by which we could more or less eliminate variations due to general causes, such as changes in the materials used, the weather, etc.

The barrows used are shown in dotted lines on the figures, and the same pattern of barrows was used in all the experiments at Glendon.

The material was dumped through six doors in the tunnel-head, equally spaced around the circumference.

The figures show a very marked difference between the sections through the doors and those taken on a diameter half-way between them, and, in addition to the greater proportion of ore and limestone opposite the doors, as shown by the drawings, the sidewise rolling caused the material of all kinds to be coarser between than in front of them, thus dividing the smelting-column into sectors of unequal fineness; but, in any sector, the finer stuff was found next the walls, the proportion of coarse increasing toward the center. I have previously alluded to this circumstance, and have shown the unequal wear in the lining of this furnace which resulted from it (*Trans.*, iv., 131).

Without asserting that this is the best possible method of filling, I can say that it has always given excellent results, and such as have not (to my knowledge) been surpassed, especially in the matter of regularity in the furnace-work.

Fig. 10 shows the distribution with the modified Langen charger, as first proportioned, for one "double" round of 24 barrows of coal, 12 barrows of limestone and 24 barrows of ore, the weight in each barrow being 4 cwt., and for one single (regular) round, 12 barrows of coal 4 cwt. to the barrow, 6 barrows of limestone and 12 barrows of ore 6.5 cwt. to the barrow. It shows very clearly the excessive heaping of the material in the center of the furnace when the stock is low, which consti-

tutes a serious defect in the apparatus, and also a very distinct difference in the section taken through the dumping-points from that taken between them. The general appearance is strikingly like Figs. 8 and 9; but there is decidedly less coal and more ore and limestone in the center.

The results of six months' work with this form were unsatisfactory, being, if anything, rather inferior to what had been done with Fig. 1. At the end of this time the opening in the center was decreased from 6 feet 6 inches in diameter to 5 feet 6 inches by bolting on pieces of boiler-plate (as shown in Fig. 11), partly because the fillers thought that enough material did not go to the center, but also because it was easier to make this change, for a trial, than any other. Fig. 11 (taken on another furnace because measurements are impossible when the furnace is in blast) shows the resulting distribution. The move proved to be a correct one, and at once resulted in great decrease of fuel-consumption, with increased make (which rose from 300 tons to 365 tons per week) and very great regularity. The fuel-consumption was, however, somewhat higher than the best results with Fig. 1; but this was clearly traced to general causes affecting all the furnaces nearly alike.

As compared with Fig. 10, Fig. 11 shows more coal in the center, and thus approaches more closely Figs. 8 and 9. The results were uniformly good when tried on several furnaces; but the unequal wear of the stock on the surface of the hopper and of the central opening in the bell caused a change to the size and shape which I have described in a previous paper (*Trans.*, xiii., 520). Figs. 12 and 13 give two distributions with this modification.

Several other variations were tried, but of only one of these (Fig. 14) was it possible to get measurements; and as none of them seemed to show any marked superiority over Fig. 12, this was adopted for all furnaces for the sake of uniformity, which was of considerable importance.

The remaining Langen figures call for little comment, except that Fig. 14 shows the effects of excessively large lumps of coal; Fig. 15 of filling the furnace very full; and that the irregularities in Fig. 16 were due in great part to a leak in the cylinder raising the bell, which caused it to come up very slowly, and the stuff between it and the hopper to dribble into the furnace instead of shooting in freely.

Fig. 11.

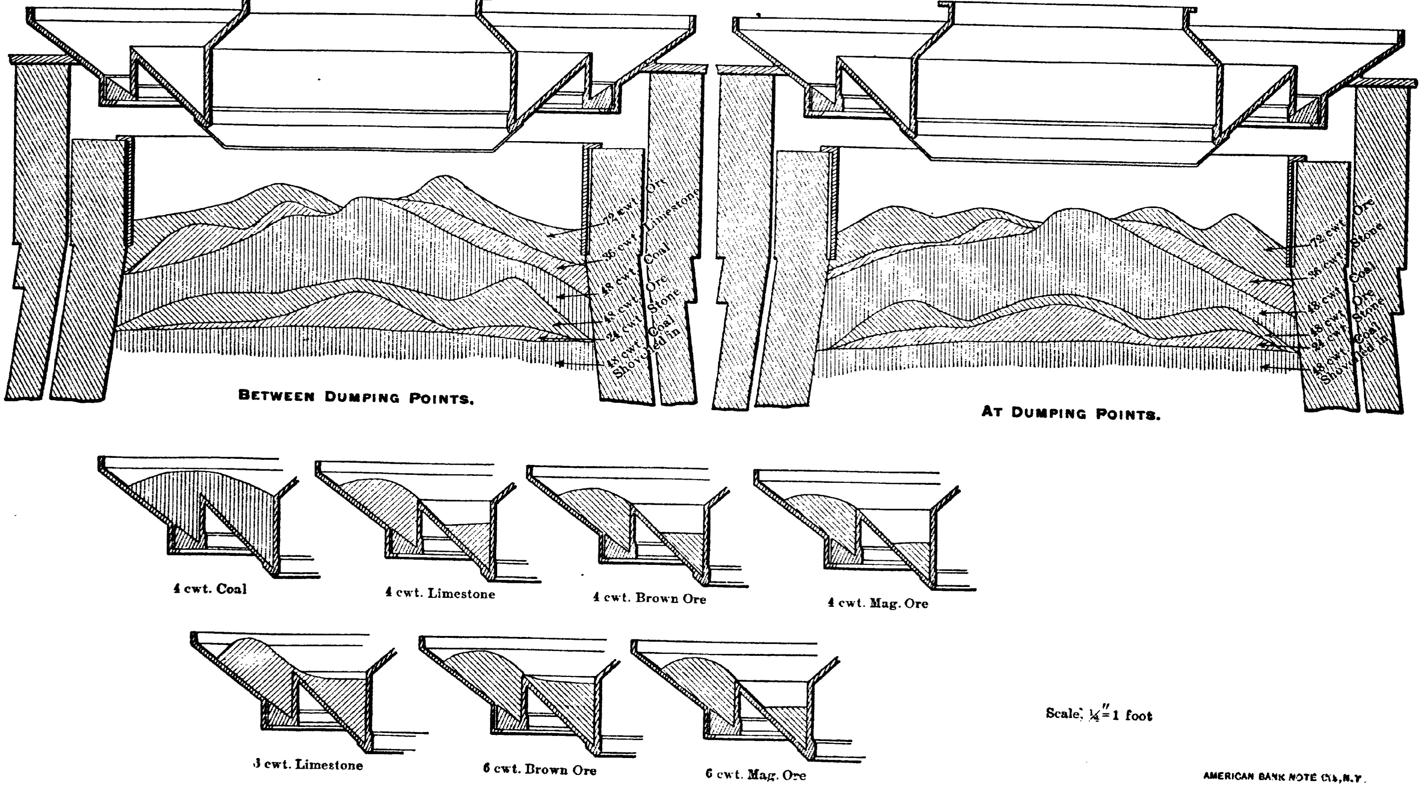


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Fig. 12.

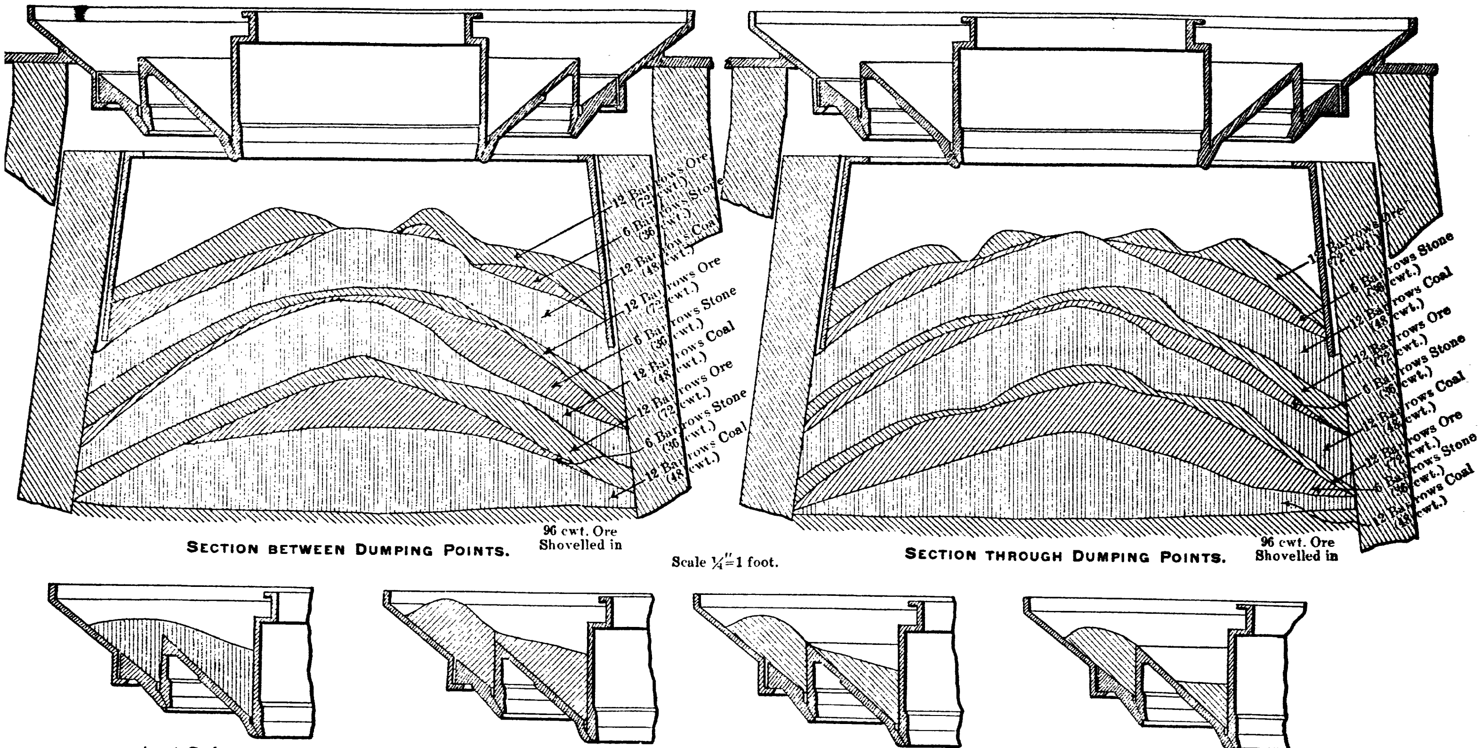
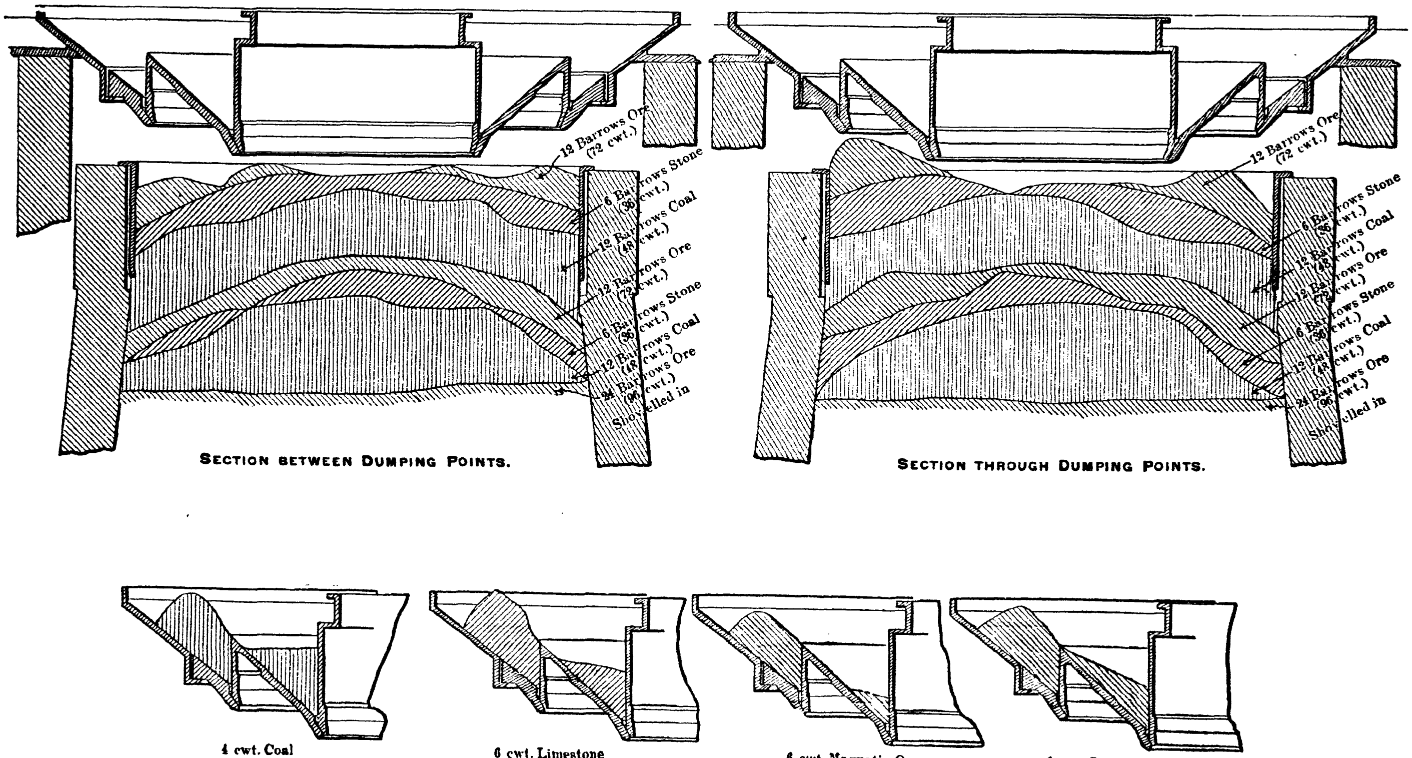

12 Barrows Ore (72 cwt.)
6 Barrows Stone (36 cwt.)
12 Barrows Coal (48 cwt.)
12 Barrows Ore (72 cwt.)
6 Barrows Stone (36 cwt.)
12 Barrows Coal (48 cwt.)
12 Barrows Ore (72 cwt.)
6 Barrows Stone (36 cwt.)
12 Barrows Coal (48 cwt.)
96 cwt. Ore Shovelled in
Section between Dumping Points.
Scale ¼″=1 foot.
12 Barrows Ore (72 cwt.)
6 Barrows Stone (36 cwt.)
12 Barrows Coal (48 cwt.)
12 Barrows Ore (72 cwt.)
6 Barrows Stone (36 cwt.)
12 Barrows Coal (48 cwt.)
12 Barrows Ore (72 cwt.)
6 Barrows Stone (36 cwt.)
12 Barrows Coal (48 cwt.)
96 cwt. Ore Shovelled in
Section through Dumping Points.
4 cwt. Coal
6 cwt. Limestone
6 cwt. Brown Ore
6 cwt. Magnetic Ore

Fig. 13.



American Bank Note Co., N.Y.

Fig. 14.

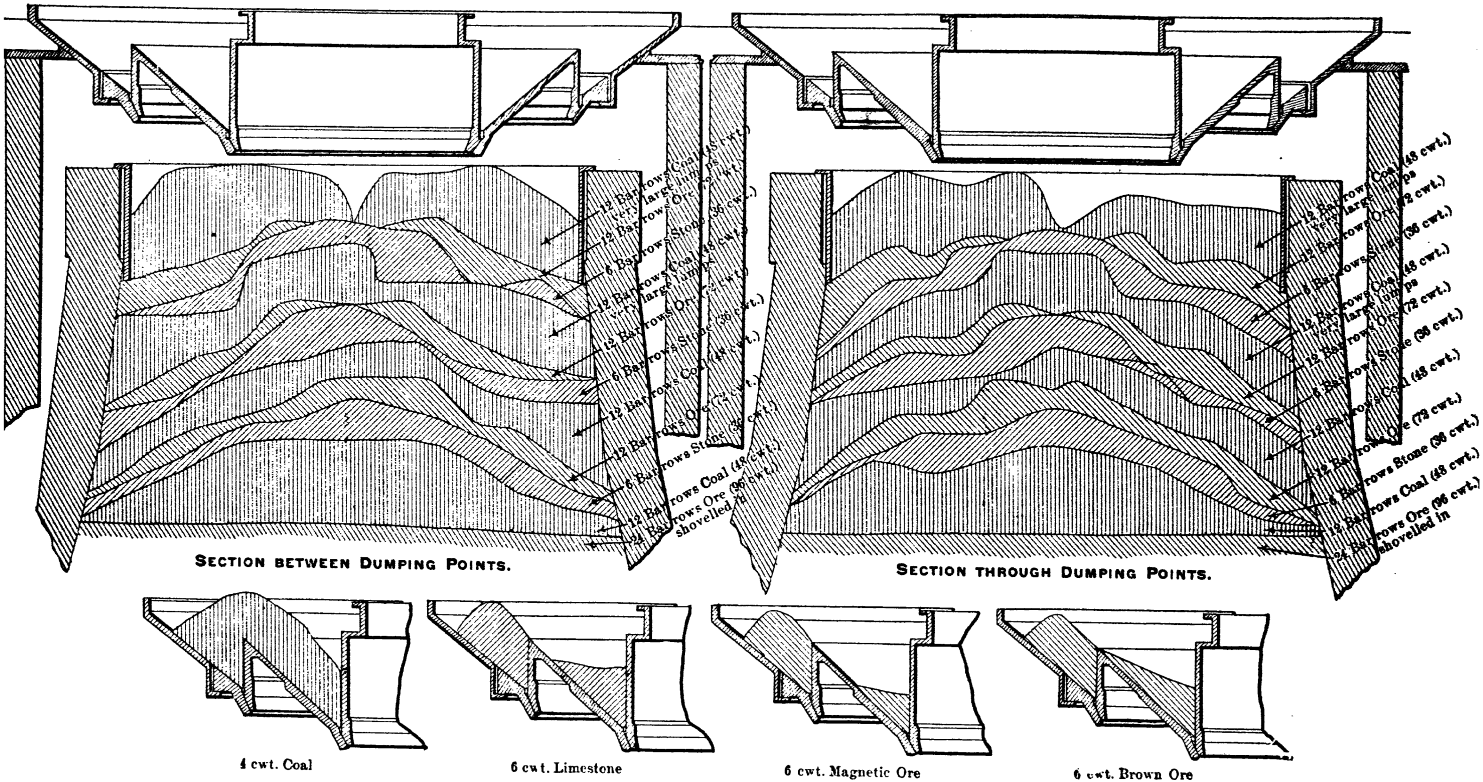
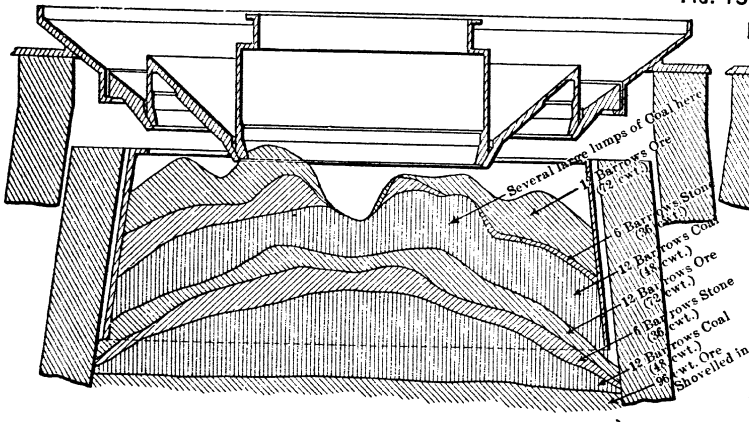
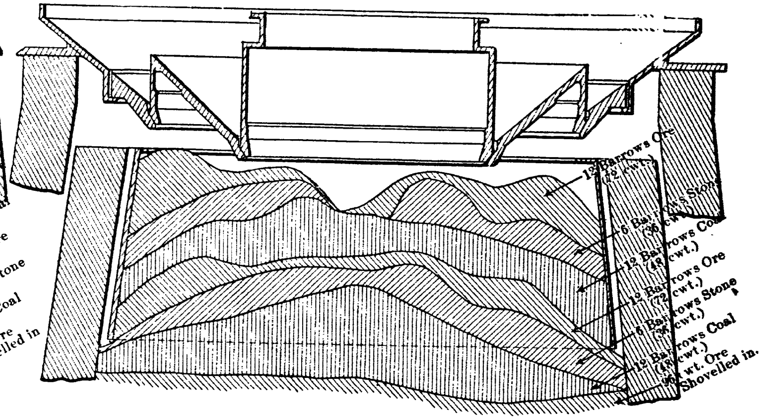

12 Barrows Coal (48 cwt.) very large lumps
12 Barrows Ore (72 cwt.)
6 Barrows Stone (36 cwt.)
12 Barrows Coal (48 cwt.) very large lumps
12 Barrows Ore (72 cwt.)
12 Barrows Stone (36 cwt.)
6 Barrows Coal (48 cwt.)
12 Barrows Ore (72 cwt.)
12 Barrows Stone (36 cwt.)
6 Barrows Coal (48 cwt.)
12 Barrows Coal (48 cwt.)
24 Barrows Ore (96 cwt.) shovelled in
Section between Dumping Points.
Section through Dumping Points.
4 cwt. Coal
6 cwt. Limestone
6 cwt. Magnetic Ore
6 cwt. Brown Ore

FIG. 15.

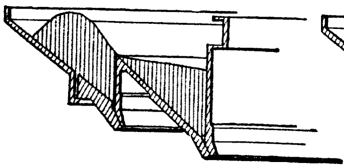


SECTION BETWEEN DUMPING POINTS.

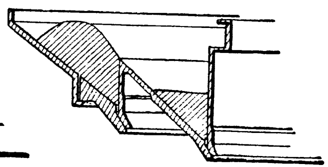


SECTION THROUGH DUMPING POINTS.

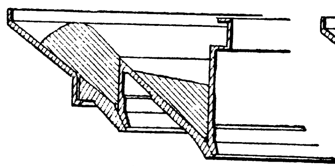
Scale ¼″=1 foot.



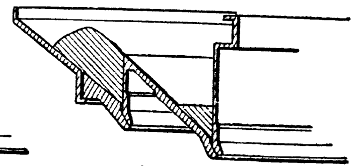
4 cwt. Coal.
(Rather small=7 to 9 in. cubes.)



6 cwt. Limestone.



6 cwt. Brown Ore.

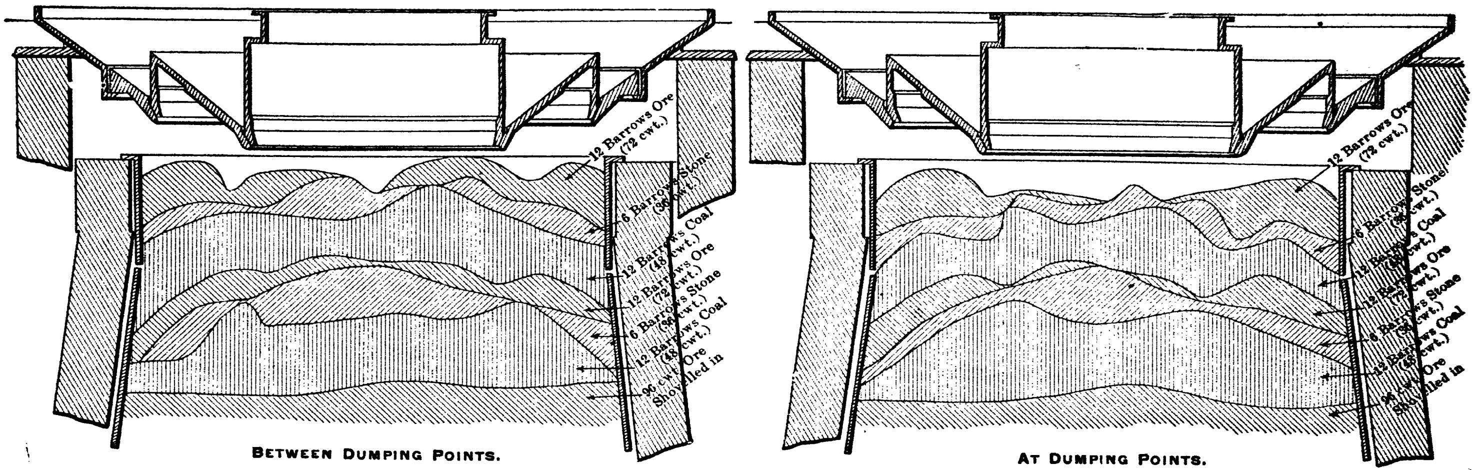


6 cwt. Mag. Ore.

NOTE:
Magnetic ore all screened over 1 in. screen before Milling.

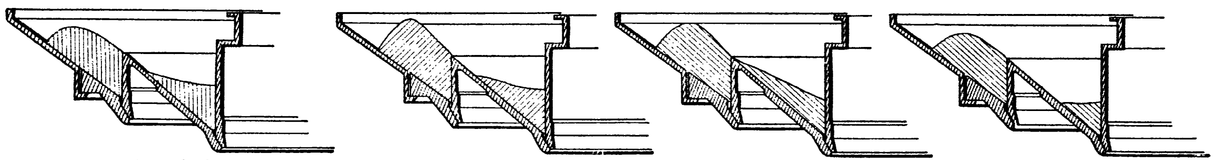
AMERICAN BANK NOTE CO., N.Y.

Fig. 16.



Between Dumping Points. At Dumping Points.

Scale, ¼″ 1 foot



4 cwt. Coal 6 cwt. Limestone 6 cwt. Brown Ore 6 cwt. Magnetic Ore

Note:

Piston in bell cylinder leaked and large bell came up very slowly.

I have had no opportunities to make like observations on charcoal-furnaces, and but few on coke-furnaces; but the latter show points of considerable interest.

The furnace on which they were made was built to use charcoal and a lean magnetic ore. When it first came under my notice the materials were hoisted on an inclined plane in cars with bottom-doors, and dumped exactly on the apex of the single bell, which had the proportions shown in Fig. 17.

With this plan of filling, the working of the furnace (on charcoal) was irregular to an extraordinary degree. Not infrequently the first part of a flush of cinder was perfectly white and spongy, but changed before the end to heavy black cinder, largely charged with iron.

It was decided to try the double bell; and as it was not possible, at the time, to alter the plane to permit the use of barrows, the charge was made to consist of four cars of charcoal and four cars of ore, the small amount of limestone needed being put into the same cars with the ore. Two cars were brought up the plane at a trip, and were dumped by a Marsaut tipping-cage, one trip to the right and the next to the left, whereby it was expected to approximate to a barrow-filling dumped at four equidistant points around the circumference of the hopper.

The work of the furnace was at once greatly improved. No measurement could be obtained, the furnace being in blast; but it was at once evident that too much ore was dumped at the ends of the cage, and, as a partial remedy, false ends were put into the ore-cars, as shown by the dotted lines on Fig. 18.*

Coke was used at the beginning of the next blast, and the distribution shown on Fig. 19 was obtained, which, as shown by the diagram *a*, departed widely from one made by barrows dumping at four points equally divided around the circumference.

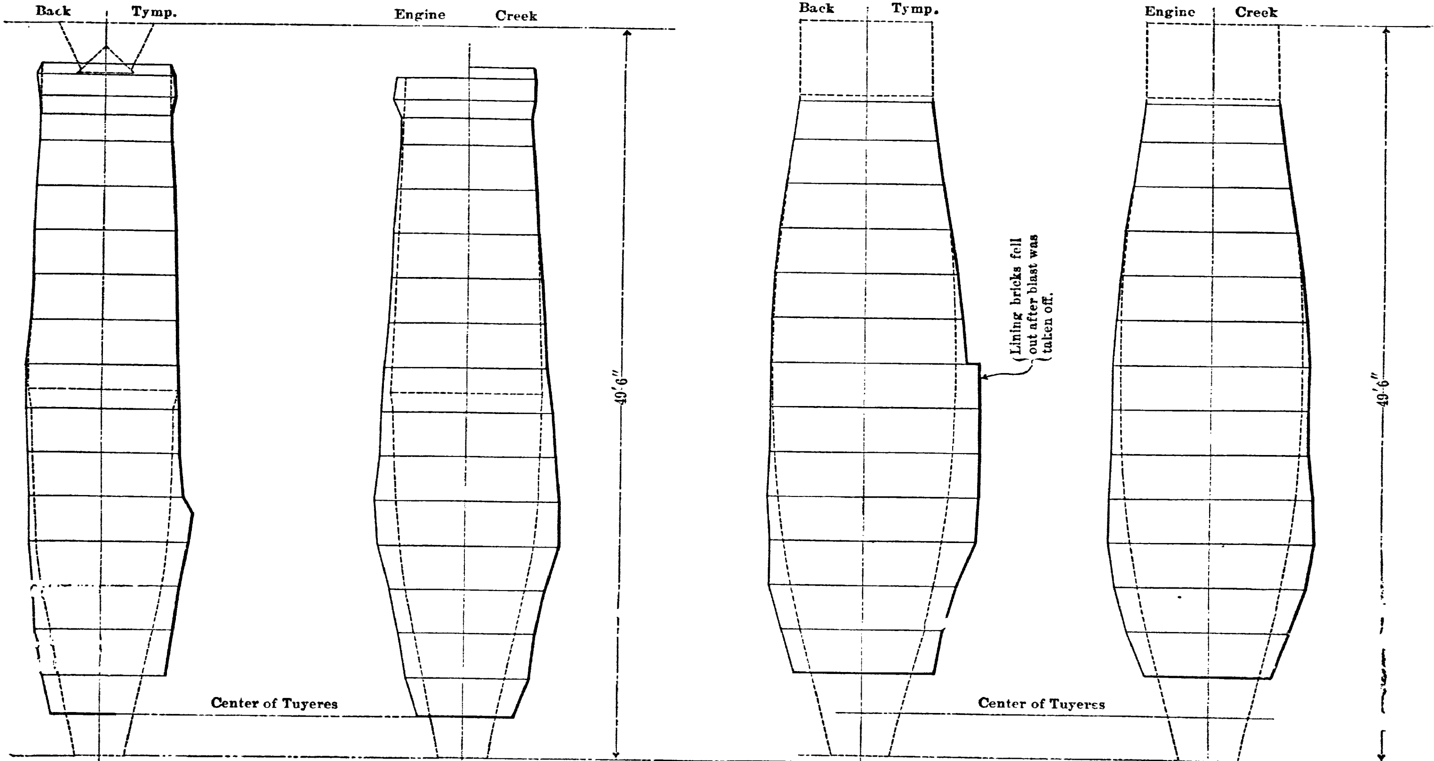
* The apparatus was improvised from parts picked up partly at Glendon and in part at Longdale, Va. The central plug was at first as shown at *a*, Fig. 18, and at once gave trouble because of large pieces of charcoal, and especially brands, remaining on it and (by getting between it and the girders from which it hung) preventing the bell from closing. To remedy this, the central plug was made conical, which at the same time increased the fraction of the charge going to the center. The furnace-work at once fell off, but was restored by putting the *fixed* cylindrical cap over the cone, as shown on the other drawings of Fig. 18.

Fig. 17 *a* shows the vertical profile of the furnace on two diameters at right angles, one of which ("Engine—Creek") passes exactly through the points at which the materials at the top were lowest, as shown by Fig. 19; and the much greater wear on this diameter is clearly attributable (as in the case of No. 4 Glendon) to the accumulation by rolling of the coarse stock on this diameter.

To remedy this, on the next blast, the bumpers of the cars were shortened as much as possible, and false ends were put into the coke-cars, as shown by the dotted lines, Fig. 18, whereby (as Fig. 20 shows) the excessive sidewise rolling (indicated by the greater thickness of the layer of ore on the sections A B Fig. 19 and A′ B′ Fig. 20, but especially by the fact that the top of each layer next the wall is there much higher than on the section "Engine—Creek") was greatly diminished. The vertical sections of the furnace (Fig. 17 *b*) show that the unequal wear, although still evident, has been greatly diminished. The Marsaut cage dumped the material *outward* from the center-line "Back—Front" on the drawings, and in consequence there was a marked tendency for the coarser stuff to roll into the outer compartment of the bell and hopper, and for the finer to remain in the middle; thus, to some extent, the material next the walls was coarser than in the center, which is undoubtedly wrong.

Finally the inclined plane was altered to permit the use of the ordinary two-wheeled filling-barrows, dumped *inward* at four points exactly divided around the circumference of the hopper, whereby the proportion of fine in the center was made smaller than with the cage-dumping. Fig. 21 shows the resulting distribution, in which the difference between the sections through dumping-points and between dumping-points has pretty much disappeared, so far as the drawings can show, although there is no doubt that the materials are somewhat coarser between than opposite the dumping-points—a matter which, as already remarked, the sections cannot definitely show. Such difference in size of material in alternate sectors of the furnace, if in any sector the stuff be, on the whole, finer next the walls than in the center, so long at least as it is kept within bounds, is favorable to good work and economy of fuel. This clearly appeared by comparing the work of No. 4 furnace at Glendon when

FIG. 17.



a

Scale, $\frac{1}{12}''$ = 1 foot.

b

Dotted lines show shape as blown in.

filled through twelve doors with the results obtained by filling through six.

The most striking difference exhibited by the drawings showing the distribution in the coke-furnace, as compared with the

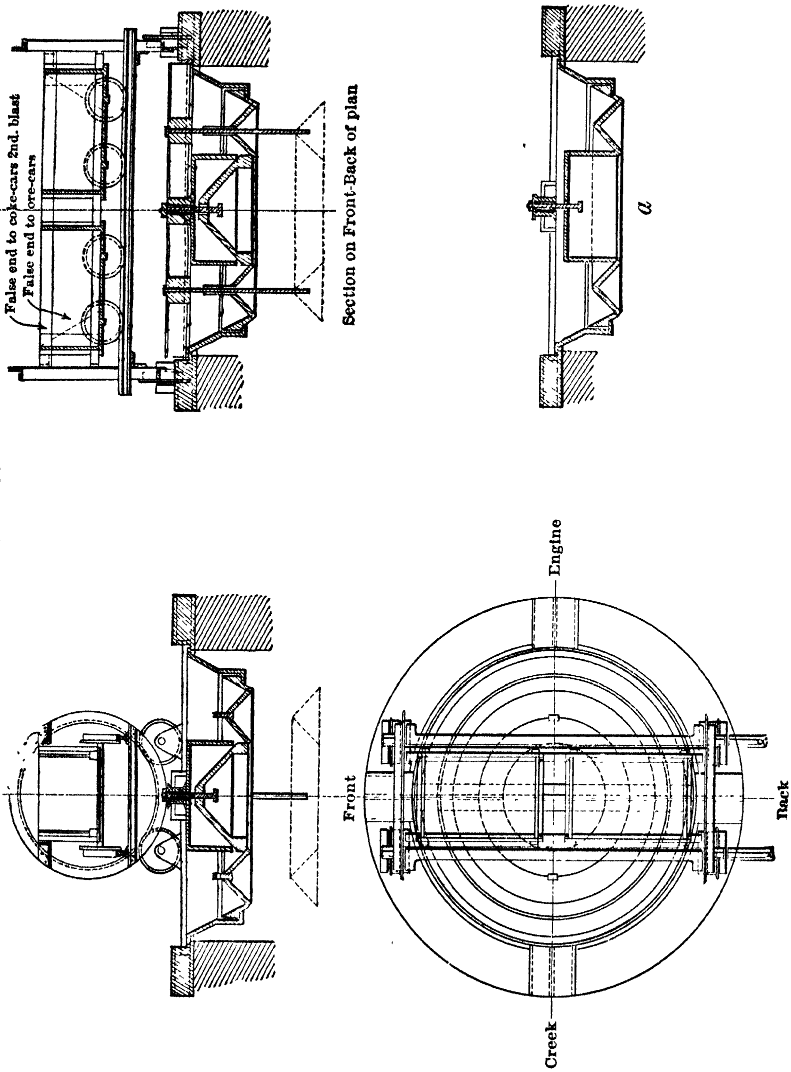
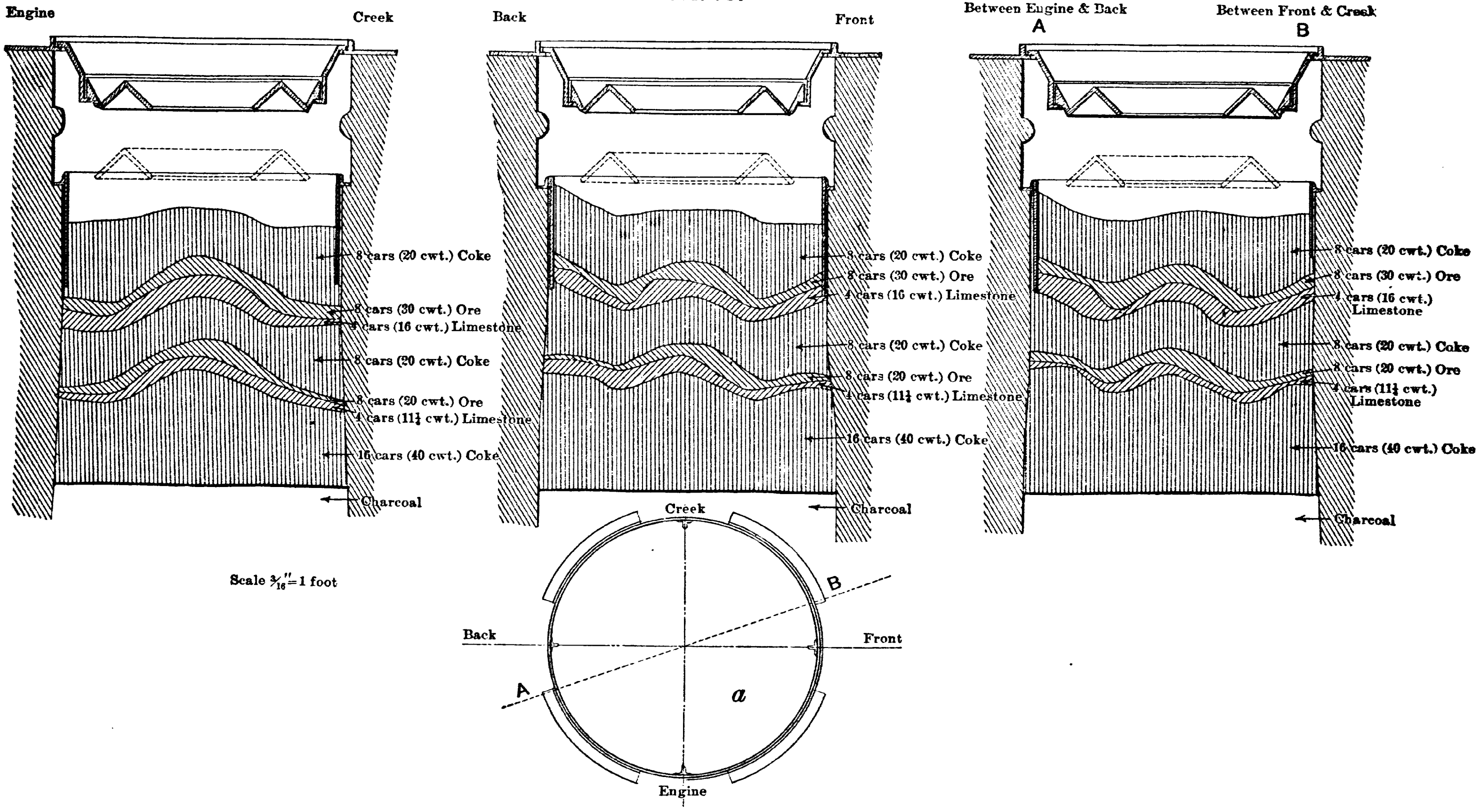


FIG. 18.

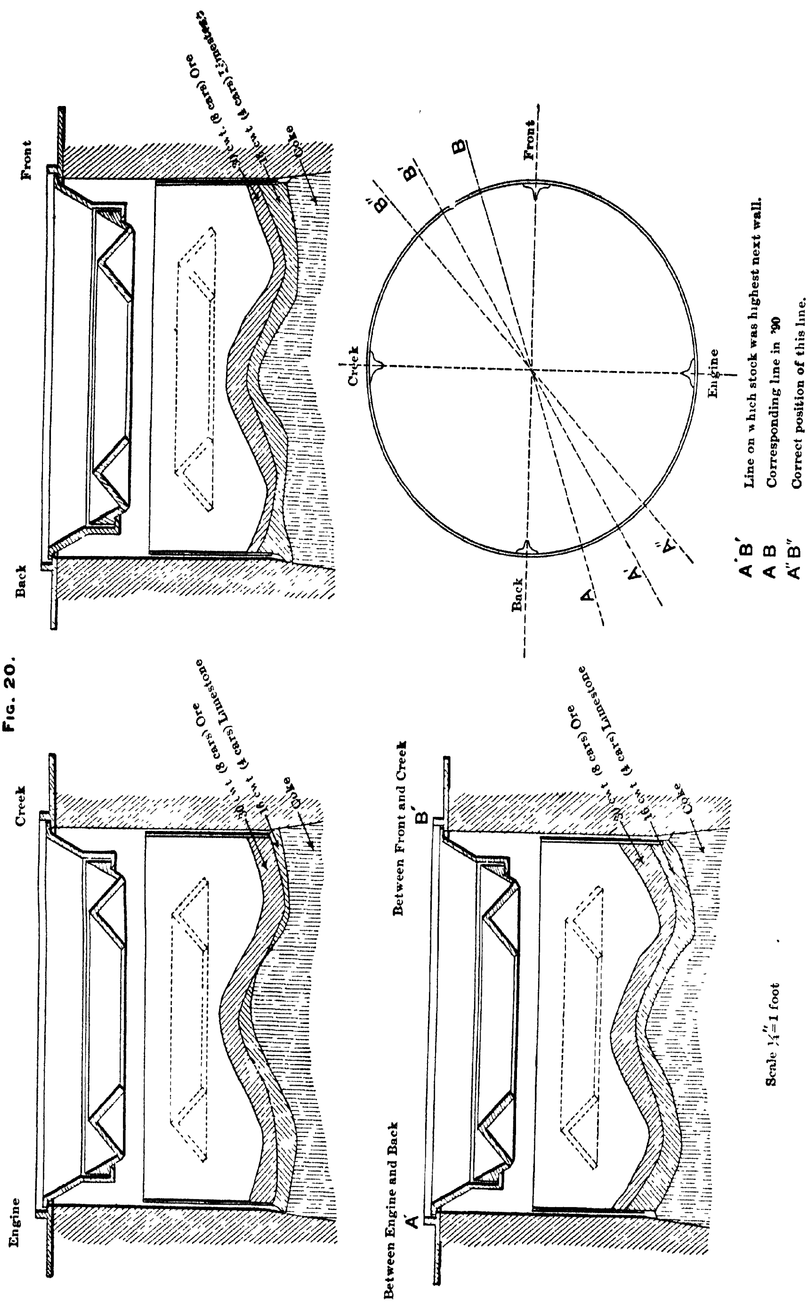
anthracite-furnaces, is in the greater thickness of the charge of coke, *i.e.*, the greater vertical space occupied by it, although the weight of the charge of coke is far smaller in proportion to the diameter of the furnace. In fact, to make the weight of the charge in proportion to the size of the furnace-top, the coke-

Fig. 19.



Scale $\frac{3}{16}'' = 1$ foot

FIG. 20.



charge should be: $10^2 : 8^2 :: 48 : x = 30.72$ cwt., whereas it is only 20 cwt.

It is also plain that the volume of the ore in a charge for the

same burden (about 1½ pounds of ore to 1 pound of fuel in the figures), as compared with the volume of the accompanying fuel, is far less in the case of coke than of anthracite. These differences and some others (all simple consequences of the fact that the apparent specific gravity of coke is (roughly) half that of anthracite) are of the greatest practical importance.

Admitting that good coke and anthracite will carry (roughly) about the same burden of a given ore, we see at once that the effect of a bad method of filling will be far more sensible with anthracite than with coke; for the greater the proportion by volume of ore in the charge, the greater the importance of having it in the right place.

The void space in any vessel filled with material in pieces of uniform size and similar shape, and with it the mean free cross-section, is independent of the size of the pieces, provided the linear dimensions of these be small compared with the diameter of the containing vessel;* but this void space may be decreased indefinitely by the addition of material in finer and still finer pieces. In most cases, the ore is charged in pieces decidedly smaller than the pieces of fuel, and in all cases of normal working it is finally reduced to dust by the action of the gas. It follows clearly that as the original void space is (roughly) the same in a charge of coke and a charge of anthracite, it will always, sooner or later, be much smaller in the charge of anthracite, simply because the volume of fine material (the ore) to fill it is much greater than with coke. The same reasoning applies in comparing charcoal with coke, the apparent specific gravities of the three fuels being approximately as 1 : 2 : 4.

The accounts of the older attempts to substitute anthracite for coke† attribute the difficulties met with to decrepitation, and the text-books of to-day very generally repeat the statement. No doubt this was, and is, a contributing cause, and in very varying degree with different varieties of anthracite; but the increased choking-up of the void spaces with ore, consequent on the high specific gravity of the fuel, is general and unavoidable.

* This is self-evident in the case of a space filled with equal spheres, and a little reflection will show that is also true for pieces of any shape, provided we consider a volume large enough to permit the elimination of accidental variations from a mean.

† *Ann. d. Mines*, 1833, iii., tome 3, p. 71 (Gueymard); *id.*, iii., tome 4, p. 127 (Robin); quoted by Walter R. Johnson, *Anthracite Iron*, Boston, 1841, p. 15.

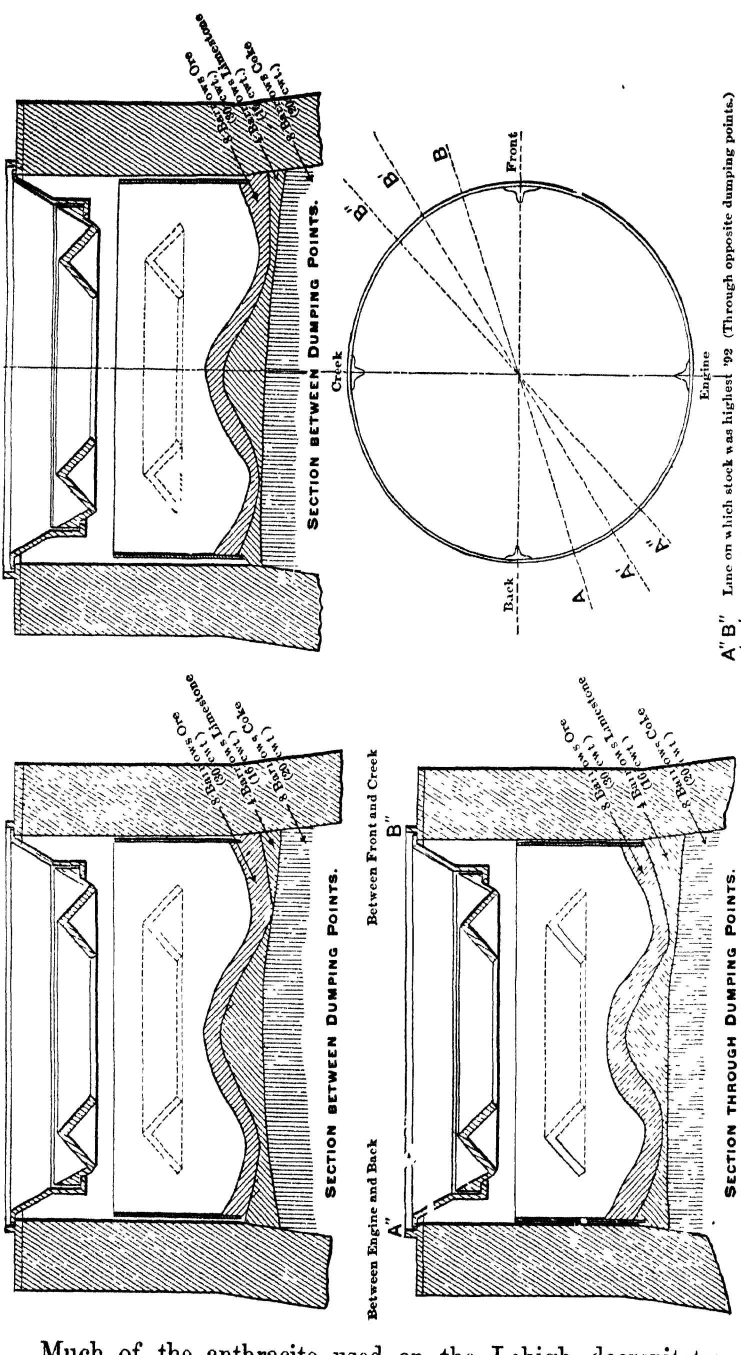


FIG. 21.

Much of the anthracite used on the Lehigh decrepitates badly when suddenly heated, as it used to be, when exposed to

the burning gas at the top of the open-topped furnaces; but in the closed-top furnaces, the temperature to which it was suddenly exposed very rarely exceeded 600° Fahr., and was almost always far lower. In spite of these improved conditions as respects decrepitation, I never observed, nor have I heard that others observed, any appreciable saving in blast-pressure as a result of changing from the open to the closed tops. Evidently, the difference between much or little decrepitation was not sufficient to overbalance the extra friction imposed on the gases by forcing all of them through the whole height of the smelting-column, instead of allowing a large part to escape through side-flues 10 or 15 feet below the top.

It is plain, therefore, that there is a simple general cause acting to decrease the void space, and thus to increase the resistance to the flow of the gases (and with it the necessary blast-pressure) as we increase the specific gravity of the fuel. This, in all probability, is the chief reason for the progressive increase in blast-pressures long ago noted in passing from charcoal to coke and anthracite. Moreover (although we cannot subject it to numerical calculation), the difference between coke and anthracite in this respect is so great as to explain readily the very sensible decrease in blast-pressure which is observed when even so little as one-eighth of coke with seven-eighths of anthracite is used in place of anthracite alone.

Bell has already pointed out* that the ore in a charcoal-furnace, because it is diffused through a greater bulk of fuel, is under more favorable conditions for reduction by the gas than in coke-furnaces, and a like comparison holds for coke as against anthracite.

The decrease in void space resulting from mixing fine and coarse materials justifies the care exercised by a past generation of furnace-managers to exclude "dirt" (fine material of all kinds, especially braize, coke-breeze and anthracite-dust) from their furnaces. It may be observed that, while fine ore may be expected to melt and run through the fuel at some point above the tuyeres, fine fuel will persist (except so much as is dissolved by the gas) until it is consumed at the tuyeres or mechanically removed from the furnace.

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* *Principles of the Manufacture of Iron*, pp. 133 *et seq.*

Modern Cupola Practice, With Special Reference to the Discussion of the Physics of Cast-Iron.

BY BERTRAND S. SUMMERS, CHICAGO, ILL.

(Buffalo Meeting, October, 1898.)

THE technologist who has devoted much attention to the foundry-business will perhaps have noticed that the present scientific development of foundry-practice is in a condition similar to that of the steel-business some twenty years ago. In many cases chemistry is looked upon with suspicion, and the old cut-and-try-methods are still in vogue, especially in the smaller foundries. The larger works have laboratories and more extensive means for applying science, and have devoted considerable time and attention to the improvement of foundry-practice. This is more especially true in the East, although in the West there are numerous firms devoting considerable attention to improving their methods through the application of chemistry to work in the iron-foundry.

It is hardly necessary to dwell extensively upon the effects of the different metalloids associated with iron, in pig- or cast-iron, in the way they are given in most text-books. The writer has thought proper to consider more particularly new developments in cupola-practice, particularly those pertinent to the discussion of the physics of cast-iron. In this connection some new ideas which recent practice has developed will be mentioned, and some results of practical tests along the same line will be described. Yet it has been deemed advisable also, by way of introduction, to outline the effects of the metalloids as they are described in most text-books, without going into detail.

The question is sometimes asked, What is the most important element governing the quality of cast-iron for any particular purpose ? It would seem that right here lies the basis for the greatest difference of opinion. Some twenty years or more ago, silicon was regarded by foundrymen as one of their worst

enemies; but since the work of Turner and of Keep, silicon has been greatly growing in favor, until one might say that it is regarded in some quarters as the panacea for all evils encountered in the iron-foundry. This has led to the founding of a school, the followers of which seem to regard silicon as an all-important element, and it would appear indeed that in some quarters it is regarded as the one element of decisive importance in pig-iron. It is the writer's opinion that graphite is the controlling element in pig-iron, and that a greater success is obtained where this metalloid is governed, than in cases where the silicon only is watched. We are more or less familiar with the well-known ideas, that by gauging silicon the exact mixture suitable for any purpose is obtained; it being only necessary to keep the carbon above a certain unknown minimum, and the silicon being determined crudely by shrinkage and other methods. Indeed, we not infrequently encounter specifications in which the silicon is specified, but no mention is made of either graphitic or total carbon.

Silicon.—This, as nearly every one will agree, is a very important element in foundry-practice, its most prominent function being that of promoting the formation of graphitic carbon, while it also lowers the saturation-point of iron for carbon. The former is the property that concerns the foundry-man almost exclusively; and it is necessary for him to know about how much silicon cast-iron should carry, in order to have the desired properties. Excluding other effects, which will be taken up later, we must always consider what kind of castings are to be made, in order to know what content of silicon to strive for in the casting. It seems fair to say that nothing absolute can be stated regarding this question. Frequently, we notice in text-books and journals, that if iron carries this or that percentage of silicon it will have certain properties. This sweeping general proposition the writer cannot accept, being of the opinion that other conditions will affect the content of graphitic carbon as much as, if not more than, the content of silicon. In general, if it is desired to make good machinery-castings of close structure, and at the same time so soft that no difficulty will be experienced in tooling them, the silicon should be between 1.50 and 2 per cent., or may even run to 2.25 per cent. It is always wise in these cases to consider the iron used

in the mixture. It is well enough to say in a general way that if we have the same chemical composition in cast-iron we shall have the same properties, but experience has shown that this is true in a general way only. For example, mixtures in which the chief component is charcoal-iron, show a perceptible difference from those made entirely from coke-iron. The reason for this seems to lie in the different conditions of the carbon, which will be mentioned later.

For light hardware, in which great strength is not a very important item, it is well to run the silicon up even as high as 3 per cent. This practice has several advantages. It tends to make the iron more fluid, so that it will take delicate moulds well, and avoid difficulty or loss due to shrinkage. It is also said to enable the foundry-man to carry more scrap in his mixture. This assertion, however, is open to serious doubt, as the amount of scrap which a mixture will carry is dependent upon the carbon-content of the mixture and the scrap.

As silicon approaches or exceeds 3 per cent., the casting becomes more and more brittle, and it is desirable to keep well within the limit of 3 per cent. in most classes of foundry-work. For most foundry-purposes, on the other hand, a silicon-content below 1.5 per cent. is to be avoided.

As stated above, these rules as to silicon are only true for ordinary foundry-mixtures. Special metals, and mixtures containing peculiar irons, are often found to contradict some of the above statements. The figures given, however, are an excellent guide for general practice and will be found applicable in most cases. It will be developed later that there are other influences which affect the iron more than does the content of silicon.

Sulphur.—The good effects of silicon are frequently counteracted by the presence of sulphur, and mysterious troubles encountered in the foundry are not infrequently traceable to this element; the practical foundry-man being, in this respect, usually at the mercy of the pig-iron manufacturer. Most brands of soft iron, both charcoal and coke, are usually considerably below the danger-limit in sulphur. Foundry-men, however, are frequently misled by the statement that sulphur need not be considered, as it never occurs in pig-irons in sufficient quantities to affect the results of their work. If analy-

ses are made of drillings from castings that give trouble to foundry-men not possessing means for determining sulphur, the trouble will, in many cases, be traceable to this element. In most instances, sulphur is introduced into the mixture through the ferro-silicon irons used to supply the silicon to the charge. The writer has found well-known brands of ferro-silicon containing 0.17 and 0.18 per cent. of sulphur, and in one case 0.34 per cent. If a careful watch is not made of these irons, or if strict specifications are not drawn upon them, it is not infrequent that 0.10 per cent. of sulphur is found. Most furnace-men endeavoring to do so can keep the sulphur below the limit of 0.05 per cent.

Not long ago, the writer was consulted in regard to some faulty castings, and analysis of them showed 0.12 per cent. of sulphur. The iron was full of small cracks radiating in all directions; but they were not perceptible on the surface, and could only be discovered when the castings were struck with a hammer. When this was done, the casting would fall into many pieces, and the fracture would plainly show that the casting had been previously cracked from one-fourth to one-half of its depth, the remainder of the shell being severed by the blow. Furthermore, the casting showed signs which might have been taken as indicating cold-shortness. In all probability, the sulphur had been introduced in this case through the ferro-silicon, as the casting carried a considerable content of silicon. A maximum limit of sulphur for good foundry-practice should be fixed at 0.10 per cent.; but foundry-men should strive to keep it below 0.08 per cent.

Phosphorus.—This element is present to a greater degree than is commonly supposed in most cast-irons. It may be said that for the greater part of foundry-work it is an excellent ingredient up to a certain limit. This limit, for most cases, is about 1 per cent. Where great strength and resiliency are desired, the phosphorus should be very much below this point. However, in cases of this kind, it will seldom be noticeable in high graphitic iron, where it is kept below 0.5 per cent. In snap-flask work, phosphorus is a very desirable element to the foundry-men, tending to make the metal fluid and to keep it so. Its effect on strength is not immediately discernible, if the test-bar of cast-iron is subjected to a transverse stress. In

most cases, if the stress be applied gradually, a high-phosphorus iron will register nearly as high as one of lower content. A severe blow upon the bar, however, will soon make apparent the difference in the irons. It is weakness under sudden shock that phosphorus most distinctly promotes; and for this reason it should be kept low, where this property would be a detriment. Like sulphur, phosphorus occurs in large quantities in the ferro-silicon irons commonly used in foundry-mixtures. It may be stated, that most of the common brands of these compounds on the market carry 1 per cent. or more of phosphorus. The writer has known cases in which 7 per cent. silicon-iron carried more than 1.60 per cent. of phosphorus. Of course, this is unknown to most foundry-men not having facilities for analyzing their iron; and in this way phosphorus is frequently introduced into mixtures which never should carry a high percentage of it.

It may be interesting to note here the case of a special semi-steel, mixed by the writer, which carried a trifle more than 0.8 per cent. of phosphorus, graphitic carbon being 1.84 per cent., total carbon 1.97 per cent., and sulphur normal. The bar broke under transverse stress at 3940 pounds. Another one made from the same mixture broke at 3910 pounds. The writer has frequently found bars made of this metal, which carried over 1 per cent. of phosphorus and broke above 3000 pounds. In the case of the 3900-pound bar, the resiliency was considerable, the bar possessing spring and showing a considerable deflection before breaking. It would seem from this that phosphorus is not nearly as much to be feared as is commonly supposed.

Manganese.—Very little has been said of manganese in its relation to general foundry-practice. It is, however, an important element in many ways. It will scarcely be noticed in a mixture, up to 0.8 per cent., as far as the ordinary foundry-man is concerned. Frequently an excellent casting is found to contain over 1 per cent. of manganese. It is not very long since it was suggested that manganese is a very beneficial element in cast-iron, and it has been asserted by some metallurgists that a considerable content of this element is desirable. The writer's experience with manganese in general practice has not been extensive, but it seems to him that this modern view is to be looked upon with favor. Although it is well known that man-

ganese promotes combined carbon, silicon predominating in the iron would tend to counteract this effect. Manganese, however, has considerable effect on the magnetic properties of the iron, which have been discussed in another paper by the author (*Journal of the Society of Chemical Industry*, xvi., 999, Dec., 1897).

It is well known that manganese possesses the property of preventing sponginess of the metal, or blow-holes, by reacting with the occluded gases of the metal; and it seems that the modern view is going to work considerable advance in this way. If the foundry-man should be able to run sufficient manganese into his mixture, without hardening the metal, to rid the iron of blow-holes or prevent sponginess, manganese will soon be in high favor for general foundry-practice. There is reason to believe that this will come to pass at no very distant date. Since the publication of the above-mentioned article of the writer, later studies of the effect of manganese upon the magnetic properties of iron have adduced striking arguments, tending to substantiate the ideas advanced by Mr. R. A. Hadfield, as to the probable existence of a carbide of manganese, which exerts considerable influence on the properties of steel. It would seem, however, that in high-carbon irons, particularly in those carrying a considerable proportion of graphite, this effect is somewhat hidden. It is more noticeable as the content of graphitic carbon decreases.

Carbon.—The consensus of opinion seems to be that a great deal is yet to be discovered relative to carbon. Anomalies have frequently been encountered in foundry-practice which seemed to indicate strongly that a clearer knowledge of the state of the carbon would greatly aid matters. There is no question that graphitic carbon is the softening agent in cast-iron; and, so far as silicon can control this, it is the governing agent. The writer hopes to show, however, that, in many cases, silicon is powerless to effect this change in the state of the carbon. It is doubtful whether the form of carbon, usually called graphite, is always composed of the same variety of carbon. It will be remembered that some five or six years ago, Prof. Ledebur, in an excellent paper on carbon in iron, described four states of this metalloid. The carbon, in a transition state toward graphite (as we may describe it), he termed graphitic temper-carbon. He also stated that there was no known method of determining this

form, and that it was always estimated with the graphite. It is open to question then if, in many cases where we determine graphite, some of it does not represent graphitic temper-carbon.

To digress a trifle, mention may be made here of an old dispute between the practical man and the technologist as to the value of the fracture in determining the quality of pig-iron. Probably most technologists will agree that, as a general indication, this practical method is very useful, but very misleading. Some time ago, the writer had an excellent opportunity to observe this. A very open-grained iron was found to contain much less graphite than an iron possessing a fracture of inferior appearance.

S. H. Chauvenet has given some interesting figures, showing that a lower bed fed from the blast-furnace is normally always close-grained, though it gives practically the same analysis as the rest of the cast; and he shows, further, how obstructions in the tap-hole which cause intermediate beds to fill up slowly will make those beds close-grained, while, if the lower bed be filled rapidly, it will show an open grain. Finally, if we compare a charcoal-iron with a coke-iron, it is a well known fact that although the charcoal-iron may carry a very much higher percentage of graphitic carbon, it shows a closer and denser fracture, while the coke-iron has a very open fracture, the graphite occurring in nests, as it were, and falling to the ground in flakes when the iron is handled. A possible explanation of this might be, that the graphite commonly determined in charcoal-iron represents relatively larger proportions of graphitic temper-carbon, while the coke-iron carries a very much smaller percentage. The iron, assimilating the graphitic temper-carbon, makes a close and homogeneous metal, while the graphite tends to open the grain and segregate, thus frequently causing sponginess in the metal. This would seem to explain the reason why charcoal-iron makes a denser and more uniform metal when introduced into foundry-mixtures, and hence is generally preferable to coke-irons. This suggestion is offered tentatively; the writer not being able, as yet, to identify the temper-carbon in such irons.

It may not be amiss, in this connection, to cite an incident in the use of the semi-steel mentioned above. This metal possesès a magnetic permeability midway between cast-iron and

cast-steel, being a gain of about 50 per cent. over cast-iron. A test-bar from a machine, which had shown inferior results, proved the metal to have the same permeability as cast-iron. This being unexplained, an analysis was made of the bar, with surprising results. The analysis was almost identical with that of a bar which had shown superior results in this direction. The graphitic and total carbon were practically identical. The mixtures were entirely the same, and the metals had been worked in the foundry in the same way, with the exception that different blast-pressure had been used. The bar showing the lower results was very open, and showed a dark-grained fracture. The bar giving higher results presented a characteristic silvery appearance, and was soft and tough. It was exceedingly dense and turned more like steel, while the former bar turned just as cast-iron would do. The micro-photographs of these two bars showed radical differences, and seemed to confirm the view that these were due to temper-carbon. Further, this metal, when properly made, is homogeneous throughout the casting, and the graphitic carbon shows no tendency to segregate. It would seem, from this, that the state of the carbon was the governing factor; and cases such as this have prompted the above suggestion.

In this laboratory an attempt was made to oxidize the temper-carbon by prolonged treatment with fuming nitric acid under high temperature, but without avail. This is in accord with previous work along this line. Hopes are entertained that investigations now in progress will yield some method by which temper-carbon may be identified and estimated in these irons.

Combined Carbon.—It is usually admitted that combined carbon embraces more than one kind of carbon. This is, to some extent, substantiated by work upon the magnetic permeability of metals relatively high in carbon. However, the present state of the art will not permit much to be said with certainty in this direction. Reference may be made in this connection to recent work of Messrs. Donath and Haissig on silicon-irons, in which they cite the fact that a high-silicon iron, when analyzed for carbon by ordinary methods, gives about 1.36 per cent. less of carbon than when the drillings are oxidized completely by combustion with lead chromate, or volatilization in chlorine. Lower results were also obtained when the metal was oxidized

with chromic and sulphuric acid. This difference, Messrs. Donath and Haissig suggest, is due to some silico-carbide. The writer has endeavored to duplicate these results, working on a ferro-silicon containing about 7.5 per cent. of silicon. The total carbon, obtained by solution in a double chloride and the residue burned in a combustion-furnace, was 2.24 per cent. In every case in which the carbon was determined either by direct combustion with lead chromate, or the residue from the chlorine treatment burned in a combustion-furnace, the results agreed quite closely with those obtained by solution in a double chloride. It is possible, however, that the results obtained by these scientists may be true for ferro-silicon higher in silicon; or it may be that the metal used for their experiments was in some respect anomalous.

Relation of Silicon to Graphite.—Having discussed these general relations, let us now endeavor to see what grounds there are for the assumption that silicon is the governing factor in cast-iron. If we have several pieces of cast-iron, made at different dates from practically the same mixture, the analyses of which show practically the same total carbon and a variation in silicon, we have an excellent opportunity to trace the effect of silicon. For example, in the following table are irons which would seem to show this quite clearly:

TABLE I.—*Analyses of Charcoal-Iron Castings.*

	I.	II.	III.	IV.
Silicon,	2.20	2.66	2.92	2.41
Graphitic carbon,	2.92	2.93	2.77	2.98
Total carbon,	3.44	3.48	3.41	3.42

The other constituents of these irons are nearly the same, and all of them are controlled as far as possible in foundry-practice. They were made from almost identical mixtures, as is clearly indicated by the uniformity of total carbon. It is evident that this table does not support the unqualified assertion that an increase in silicon causes a proportional increase in graphite, and the practical rule, based on that theory, that silicon may be blindly added to the foundry-mixture, without considering other conditions, in order to increase graphite and soften the iron, is not substantiated. Comparing irons I. and III., we find that although the latter contains 0.72 per cent. more of

silicon than the former, the graphite is 0.15 per cent. lower, while the total carbon is practically the same. Obviously this increase in silicon has not produced graphite, yet the total carbon indicates that there was no marked difference in burden.

Taking IV. as the second in the series, we see that silicon increases by about 0.20 per cent., yet that the graphite is nearly constant, except as to III., which is both highest in silicon and lowest in graphite. The fair deduction seems to be, either that silicon has no marked effect upon graphite in ordinary foundry-practice, or that there are other conditions more potent. The latter view seems the more probable.

The above analyses are taken from a vast number made in the course of practice, which confirm this conclusion. Daily records for months show conclusively that the silicon varying between 2 and 3 per cent. has not nearly the effect on the graphite that it is usually supposed to produce. No relation apparently exists between the change of silicon and the content of graphite within these limits, and there is little if any doubt, if we can judge from this long series of tests, that there are other influences in cupola practice which are more potent than the variation in silicon. The above table has been selected as most clearly setting forth this view.

Another proof of this proposition is seen in the analyses of pig-iron, before they are introduced into the cupola. Pig-iron shows the effects of the metalloids in the blast-furnace, instead of in the cupola; yet even in the blast-furnace, with its high temperature, the silicon is not always able to govern the graphite. For example, the following are strong indications in this direction:

TABLE II.—*Analyses of Pig-Irons.*

	I.	II.	III.	IV.
Silicon, ...	7.94	7.43	3.36	3.30
Sulphur, ...	0.041	0.029	0.051	
Phosphorus, ...	1.39	1.05	0.606	
Graphitic carbon, .	2.02	1.95	3.31	3.26
Total carbon, . .	2.24	2.19	3.33	3.37

Here it appears that the irons carrying more than 7 per cent. of silicon have less graphite in proportion to the total carbon than the one containing 3.36 per cent. Number III., however, is a remarkable iron, and one that is seldom seen, although

its high proportion of graphite to total carbon is quite characteristic of the brand, and has been found in most shipments received from this furnace. This anomaly must therefore be due to local conditions.

The castings in Table I. were all compounded from high-graphitic charcoal-iron. In coke-iron mixtures the failure of silicon to increase the percentage of graphite is even more marked. The following analyses are taken from casts made entirely from coke-irons:

TABLE III.—*Analyses of Coke-Iron Castings.*

	I.	II.	III.	IV.	V.	VI.	VII.	VIII.
Silicon,	2.85	3.76	2.62	2.47	3.18	3.11	2.79	2.95
Sulphur,	0.073	0.083	0.074	.....	.....	.....	.....	.....
Phosphorus,	0.557	0.612	0.469	.....	.....	.....	.....	.....
Manganese,	0.39	0.260	0.42	.....	.....	.....	.....	.....
Graphitic carbon,	3.13	3.05	3.17	2.55	2.69	2.78	2.67	2.61
Combined "	0.18	0.24	0.08	0.74	0.51	0.42	0.54	0.60
Total carbon,	3.31	3.29	3.25	3.29	3.20	3.20	3.21	3.21

The first four members of this table have about the same total carbon, and were made from the same irons, with but minor changes in the burdening. It is readily discernible from these irons that little if any relationship can be traced between the silicon and the graphitic carbon. Looking at the last four members, the total carbon contents are seen to be practically identical, while the graphitic carbon is very nearly the same in the irons containing the highest and the lowest content of silicon; and, again, no relation can be traced between silicon and graphitic carbon. As in the case of the first four, these mixtures were compounded from the same irons with but minor changes in the burdening.

That temperature controls the effect of silicon is shown when a close examination is made of different shipments of the same brand of pig-iron, the blast-furnace running on the same ore-mixture. A case came under my personal observation recently, in which the graphitic carbon ran from 3.40 to 3.50 per cent., the silicon remaining in the neighborhood of 1.5 per cent.; but when the silicon rose to 1.8 or 2 per cent., the graphitic carbon rose above 3.50 per cent. In nearly every case of this kind a decrease in the total carbon was likewise noticed. This has also been frequently noticed with other brands of iron, and it is more

apparent with charcoal-irons than with coke. The heat in the blast-furnace seems to be great enough to make the silicon a more important factor.

It is hardly legitimate to attribute these differences to the effect of the other elements, as a glance at the tables will show that they do not vary enough to enter into the consideration. It would seem, on careful consideration, that the effect of silicon is largely governed by the temperature at which it is allowed to act. It is doubtful if the necessary temperature is obtained in the cupola to permit the silicon to have a very strong effect upon the carbon, where its content does not vary beyond certain limits. It also seems probable that, where the burden is light, the effect is not as marked as in the case of large burdens, where more heat is developed.

Influence of Coke-Ratio.—Frequently one hears among foundry-men that this or that one uses certain fuel-ratios, but it is seldom stated under what conditions the melt is made. One foundry-man is melting iron with a ratio of one to thirteen, while his neighbor is running on a ratio of one to seven. This means almost nothing, unless the amount of iron melted and the condition of the material introduced into the cupola are taken into consideration. It need scarcely be said that the foundry-man who melts 60 tons per day can make a better showing in relation to fuel-consumption than one who melts 5 tons, and has the same iron poured from his cupola. The foundry-man having a smaller burden uses more coke on his bed in proportion to the iron melted. It is also evident that, where it is desired to melt large pigs and large-sized scrap, a greater coke-consumption is necessary, to tap the same iron from the cupola. Barring these two conditions, which are evident to every one, the iron is, within certain limits, softer when more coke is used. If analysis is made of metals cast with different coke-consumption and practically the same burden, as a general rule, the one melted with the higher coke-consumption, up to a certain limit, will contain the most graphite. This is only true when the cupola is run on a coke-consumption below what is really necessary to pour iron of a high temperature, and this, again, is largely dependent upon the blast; but where little attention is devoted to the blast, and especially where the blast-pressure is below what it really should be, an

increased coke-burden will often have a beneficial effect upon irons, if too much is not added. If a foundry-man is using his blast judiciously and the coke-ratio is figured for the best economy, little can be gained from an increased coke-consumption.

Influence of Oxidized Material, Especially Rusty Scrap, upon Mixtures.—Little attention has been paid to the peculiar effect of introducing rusty material into the cupola. Only one experience of this kind is necessary to convince a foundry-man of the deleterious effect of corroded scrap. The surface-rust on pig-iron is usually not noticeable, but where corrosion has taken place to any considerable depth, a very bad effect is quickly discovered. This is particularly true where light scrap is used, and a large surface has been exposed to the corrosive effect of the atmosphere, in proportion to the small volume. A prominent effect in bad cases of this kind is to make the iron exceedingly dirty and spongy, and when this dirt is noticeable, the cleaner the material introduced into the cupola, the cleaner will be the casting obtained. The writer was confronted some time ago with a serious complaint of dirt in the iron, and on investigating the case, found it due entirely to light scrap which had been seriously corroded. Upon the dropping of this scrap from the mixture, the trouble entirely disappeared. There is very little doubt that, in many cases, spongy iron poured from the cupola is traceable to the use of corroded material. Besides this inconvenience, material of this nature tends to harden the iron to a limited extent. But this is not nearly as noticeable as the former effect, and cannot be detected in iron high in graphitic carbon.

Effect of the Blast.—There is scarcely any one factor in cupola-practice deserving of more close and constant attention than the blast. There are reasons to believe it exerts considerable influence in furthering the mutual reactions of the different elements in the metal, but this has not been clearly demonstrated in the tests thus far made. Aside from its effect in this way, increase of blast seems to have a decided tendency to increase the total carbon in the cast-iron, and correspondingly to increase the percentage of graphite. As demonstration of this, the composition of the irons in the following table, which were made from the same mixture at different dates, may be interesting:

TABLE IV.—*Analyses of Castings from the Same Mixture.*

	I.	II.	III.	IV.	V.	VI.	VII.
Silicon, ...	2.30	2.74	2.03	2.27	2.04	2.10	2.21
Graphitic carbon, . . .	2.98	2.59	2.98	3.07	2.90	3.11	3.06
Total "	3.44	3.15	3.67	3.52	3.61	3.74	3.63
Lbs. of iron per lb. of coke,	6.4	5.4	6.2	6.3	7.6	7.6	7.0

The first three members of this series were cast from a larger cupola than the remaining members. Only the third iron shows a high carbon content that compares with the last three. The blast apparently varies. Iron is sometimes poured, which can only be accounted for in this way. If II. is contrasted with I. and III., a decided difference is noticed; also III. differs considerably from I. and II. These irons are all made from practically the same mixture, and are here mentioned to show the anomalies that are so common in foundry-practice. Unfortunately, reliable measurements have not been secured on blast-pressure, but, inasmuch as the air is supplied by an open blower, the amount of air passed through the cupola would be largely dependent upon the resistance found in the cupola itself. The true criterion of the effect of the blast is the amount of air passed through the cupola in a given time, and not the blast-pressure. Whenever an attempt was made to measure the blast-pressure, it was always found to be practically the same, even when both cupolas were run at once. The explanation of this is obvious. Inasmuch as the fan is not entirely enclosed, the pressure can never exceed a certain maximum, or air will be forced back through the fan. In the case here cited, the fan was somewhat overworked, and probably air was being forced back through it. It would thus happen that, if the burden offered more resistance at one time than at another, less air would pass through the cupola at that time. This would seem to be the explanation of the difference in these three irons made from the same cupola.

This hypothesis is apparently verified when the remaining four members of this series are considered. These four irons are compounded from the same pig-irons as the first three, as are also all the irons in Table I. The mixture has been varied but slightly, and if any difference could be expected from this cause, the irons in Table I. should show higher carbon contents than those of Table IV. The last four irons of Table IV. were

melted in the same cupola as those of Table I., but more air was passed through the cupola in the former case. Also more air was passed through the cupola in melting the last three members of Table IV. than in melting No. IV. of this Table. This was accomplished by using more tuyere-openings. In the case of iron IV., two openings in the upper set of tuyeres were used, besides the full lower set. But with the last three irons of this series the full upper, with the full lower set, was used. With the irons of Table I., only the lower set was used. Contrasting iron IV. with those of Table I., a higher content of total carbon is noticed, and also a corresponding rise in the graphitic carbon. Again, comparing members V., VI. and VII. with IV., the same thing is noticed. This is particularly noticeable in VI., and is probably due to the causes mentioned above. These tables do not appear to demonstrate that silicon has had any greater effect in promoting graphite under the effect of the increased blast. It may be that if the blast-pressure were increased to a still greater extent, an effect would be noticeable. It would seem almost a demonstrated fact that the temperature is the governing factor of the effect of silicon, and the temperature of the cupola is certainly increased by increased blast, if a certain maximum be not exceeded. This limit is seldom reached in foundry-practice, and faulty iron is more frequently due to a scarcity of air than to an excess. The blast is probably one of the most important factors in foundry-practice, and many of the mysteries not yet explained may be traced to its door. There is scarcely any doubt that it has a marked effect upon the content of total carbon, and it is not unlikely that when the results of further investigation are known, the amount of air passed through the cupola will be found to condition, to a certain extent, the effect of the metalloids in the iron.

Iron-Mixtures and Iron-Specifications.—An attempt has been made in the foregoing pages to demonstrate that the desire for silicon has been carried to an extreme. This desire has become a mania in some quarters. If the foregoing tables prove anything, they certainly prove that high-silicon irons are, in many cases, a useless luxury. A certain amount of silicon is undoubtedly necessary, but the plan of gauging the value of irons by their content of silicon is but one step in advance of the old

fracture-method. The carbon is undoubtedly the governing factor in irons, and the most radical advocate of silicon can do nothing with foundry-mixtures without a certain carbon content. It seems, then, that total carbon is one of the most important elements to specify in purchasing foundry-irons. The writer has yet to meet with an iron too high in carbon to be of excellent use in the foundry. It has been the custom in our practice to specify 3.75 per cent. of total carbon in our No. 1 iron, especially when charcoal-irons are purchased, while a very lenient specification is allowed in silicon, it being, for charcoal-iron, not below 1.50 per cent. The furnace-man is allowed to have silicon about anywhere he wants it as long as he can furnish iron with the necessary carbon. A minimum graphite specification is also inserted in nearly every case. The philosophy of this is obvious from the foregoing pages. In the cupola, comparatively little combined carbon is changed to graphite, and this is especially true when the heats are small, and, consequently, the heat developed is not sufficient to produce any marked change in this direction. This fact can easily be demonstrated in practical work, where frequent analyses are made of daily casts, and the compositions of the irons composing the mixture are known.

In these specifications the minimum is specified, and any variation must be above this limit. This is thoroughly practical, and has been in operation long enough commercially to demonstrate its value. It is difficult to get iron-merchants to make carbon-determinations, but they will agree to furnish iron under specifications requiring certain minimum limits of total carbon and graphite. Whether the latter is specified or not, there can be no doubt about the necessity of specifying total carbon. Very little attention is paid to silicon in the work, as most of the high-carbon irons will carry sufficient silicon for most classes of work. However, it is often useful to require a certain silicon-content in the iron, so as to get sufficient graphite, as the heat of the blast-furnace is sufficient to enable the silicon to control the graphite to a great extent. In the best snap-flask work, where a large proportion of charcoal-iron is used in the mixture, 2 per cent. of silicon is found to make elegant castings, and in such mixtures no ferro-silicons are used; but a high-silicon coke-iron, carrying over 3 per cent. of silicon

and about the same of graphitic carbon, is used. This gives good results, especially where a high grade of machinery-castings and fine snap-flask work are poured from the same mixture. This is further substantiated by analyzing some of the best castings of tools and machines that can be found on the market. Some of the best-known firms making this class of work will be found to be using this variety of iron. With coke-iron it is well to run the silicon a little higher, although for machinery-work it is best not to have it too high.

When trouble is experienced with open or spongy iron, and the trouble cannot be traced to some such cause as rusty scrap, a good procedure is to run the total carbon in the casting as high as possible. This will give a higher content of combined carbon, and at the same time the high content of total carbon, with the graphite, usually increased slightly in this way, will keep the iron soft. When this is done it is well to consider the content of silicon, and if this is much over 2 per cent. a decrease will be found beneficial.

The Use of Carbon.—As mentioned in the first portion of this paper, silicon is commonly believed to enable a mixture to carry more scrap. This belief is open to serious doubt. It is obvious that if a mixture is running low in carbon, and ferro-silicon is added in greater amounts to enable the mixture to carry more scrap, this procedure will not only fail to enrich the carbon content, but will actually impoverish it. The reason for this lies in the fact that high-silicon iron carries less graphitic and total carbon than a good scrap, and much less than a good cast-iron. For this reason it is evident that when ferro-silicon is added to the burden, it is certain to lower the content of carbon in the cast-iron somewhat. Inasmuch as the iron is dependent upon carbon for its softness, it is open to grave doubt whether, by increasing the silicon, and thereby decreasing the carbon, by adding ferro-silicon, any more scrap can be carried, and the same quality of iron poured from the cupola. The amount of scrap that can be loaded into the cupola without changing the quality of the iron is dependent upon the carbon, and especially upon the graphite. Even if we admit that an increase of silicon can cause an increase of graphite, yet, if there is not sufficient carbon present to be changed into graphite, the graphitic carbon cannot be obtained in the re-

quired proportion. Further, when it is doubtful that silicon will cause any appreciable increment in the graphite, it is open to question if the ferro-silicon does not tend to lower the scrap-carrying ability of the mixture.

A high-carbon mixture averaging, for example, 3.10 per cent. graphite, and about 2 per cent. silicon, will carry considerably more scrap than a mixture containing over 3 per cent. silicon, and lower in graphitic carbon, or than a mixture impoverished in total carbon, no matter what the silicon-content. This assertion can be proved in practical work, and it would seem that the popular idea to the contrary is based on a fallacy.

The carbon-determinations in the foregoing tables were all made by the oxygen combustion method, and the greatest care was exercised to insure accuracy. The determinations have been checked, and the results from the furnaces used, checked with results from other laboratories. In this connection, thanks are due to Mr. E. J. Ericson, whose excellent work in this particular has greatly aided this investigation.

In conclusion, it should be mentioned again that the fan supplying the air to both of the cupolas employed in this investigation was somewhat overloaded. Therefore, it is possible that foundries supplied with better blast facilities will obtain results different from the above; that a greater effect of the silicon will be noted, and perhaps a greater benefit traced to the use of ferro-silicons. If this should be so, it would only go to demonstrate the effect of the blast in foundry-practice. It is hoped that any members who may have noted different effects from those outlined above will make their observations known.

Experiments in the Sampling of Silver-Lead Bullion.

BY G. M. ROBERTS, MURCHISON GOLDFIELDS, WESTERN AUSTRALIA.

(Buffalo Meeting, October, 1898.)

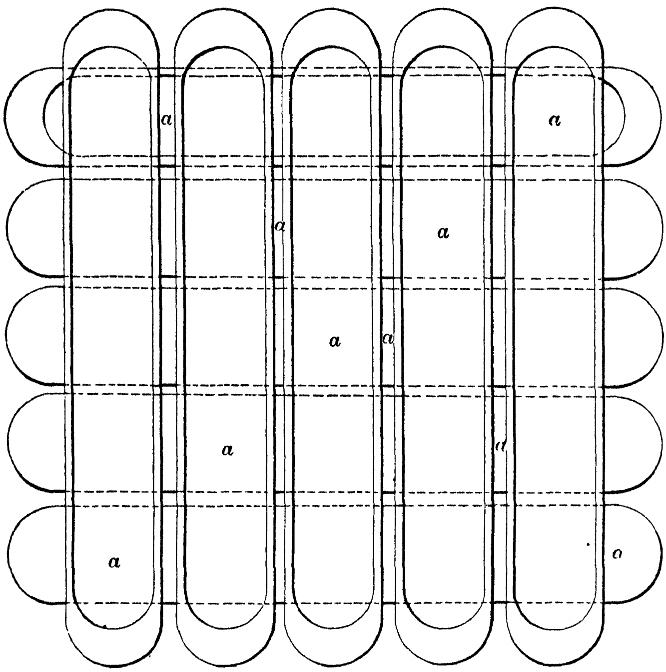
DURING the six years that the writer was connected, as chemist and chief assayer, with the Proprietary Mines, Broken Hill, N. S. W., several interesting experiments were made in the sampling of the silver-lead bullion obtained from the water jacket smelting-furnace.

The smelting-plant is divided into two portions, known locally as the north furnaces and the south furnaces.

The former group comprises nine and the latter six 80-ton water-jacket furnaces, besides which the south group has one small furnace used for smelting dross and concentrating the matte which is obtained in the larger furnaces, etc.

At the north furnaces, the lead is tapped into lead-kettles and allowed to stand for a short time, then skimmed and ladled into moulds fixed on a stand alongside the furnace. When the lead

FIG. 1.



has set, the bars are taken out of the moulds and stacked, as shown in Fig. 1. At the south furnaces, the bullion is tapped from the furnace into a pot, and, while still molten, is wheeled over and tipped into a liquation-furnace, where it is purified. When enough bullion has accumulated in this furnace to make one lot (150 bars), the lead-tap is opened, and the lead is allowed to run into moulds placed on a revolving tray, which holds 150 moulds. These are generally filled at one tapping, and hold about 5 tons of bullion, or about 80 pounds to each bar.

The writer is indebted to Mr. W. J. Koehler, chief metallurgist, and his assistant, Mr. A. E. Savage, for the different methods of sampling the bullion. The samples were all carefully taken under their supervision and sent to the assay-office. The object of these experiments was to ascertain the relative accuracy of the methods then in use at the furnaces. The following results will be interesting to readers connected with metallurgical works.

I.—Bars from the Liquation Furnace.

A lot of 150 bars, taken from the liquation-furnace, was sampled and assayed as follows:

(A) Usual method of sampling, *i.e.*, one dip-sample is taken at the beginning, one in the middle and one at the end of the run. Equal weights of these dips are taken, hammered into sheets about $\frac{1}{8}$-inch thick, and then cut into small pieces. These are thoroughly mixed, and 0.25 assay ton is weighed out for assay. All assays are made at least in triplicate. The assay result was 469.4 ounces per ton. The three dips were assayed separately, with the following results:

No.	Ozs. per ton.
1,	475
2,	472
3,	466
Average,	471

(B) As the bullion was running from the furnace into the moulds, a dip-sample was taken at every tenth bar, making fifteen samples altogether. These were hammered into thin sheets, and assays were made from each, with the following results:

No.	Ozs. per ton.	No.	Ozs. per ton.
1,	474	10,	468
2,	475	11,	464
3,	473.6	12,	468
4,	473.6	13,	466
5,	472.8	14,	465
6,	464	15,	465
7,	472		
8,	470	Average,	469.8
9,	472		

(C) Sample taken from the lugs, five bars being chipped on

the top and five on the bottom, alternately (at points *b*, Figs. 3 and 4).

Assay-result, 482.7 ounces per ton.

(D) Sample taken from the center of the bottom of the bars (points *a*, Fig. 4).

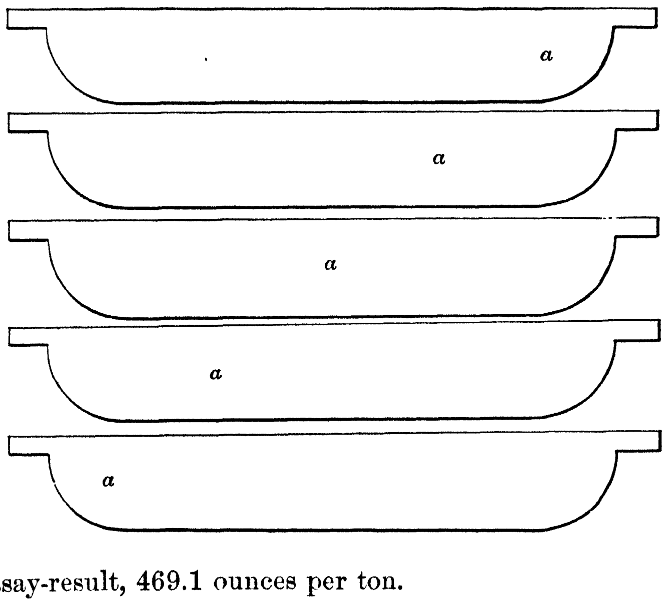
Assay-result, 471.1 ounces per ton.

(E) Sample taken from the center of the top of the bars (points *a*, Fig. 3).

Assay-result, 439.1 ounces per ton.

(F) Sample taken diagonally across the top of the bars as they were stacked (points *c*, Fig. 3).

FIG. 2.



Assay-result, 469.1 ounces per ton.

(G) Sample taken diagonally across the bottom of the bars (points *c*, Fig. 4).

Assay-result, 466.5 ounces per ton.

(H) Sample taken by sawing each bar through transversely (see Fig. 5). The saw-dust obtained amounted to about 2 cwt. This was carefully mixed, then quartered, and each quarter was separately cut down by another quartering, until four samples remained, of about 3 pounds each. These were each washed with benzine, to remove any oil or greasy matter derived from the saw. In this washing operation the samples lost 0.25 per cent. in weight. The assay-results were:

No.	Ozs. per ton.
1,	469
2,	471
3,	471
4,	469
Average,	470

Comparing the different results, we find that the average samples taken agree very closely. They are for

	Ozs. per ton.
Method A,	469.4
" B,	469.8
" H,	470.

This proves undoubtedly that the method in use, of taking the dip-samples from the bullion passing through the liquation-furnace, is perfectly correct. When this furnace was first started, the first four lots of bullion tapped from it were sampled by both methods A and B before adopting the former. In every case the results were the same. Samples F and G show that in this lot of bullion there was very little difference in value between the top and bottom of the bars; but, contrary to expectation, the top-sample goes slightly higher. Only in cases where the bullion is sampled by chips is this point important. The greatest variation occurs in samples C and E, and there seems to be nothing to indicate the reason. The top of the bar is invariably poor, and the lugs and sides are the richest.

II.—Bars from the Lead-Kettles.

A lot of 150 bars, moulded directly from the lead-kettles at the north furnaces, was sampled as follows:

(I) After a kettleful of lead had been tapped, the bullion was carefully skimmed, and a dip-sample was taken near the beginning and towards the end of the ladling. Each lot of bars thus tapped, varying from 11 to 16 bars, was weighed separately, until the whole lot of 150 bars had been made up. In the assay-laboratory, aliquot parts of the dips (corresponding to the weights of the different lots of bars) were weighed out and melted into one general sample.

Assay-result, 408.7 ounces per ton.

(J) Samples taken by the old method, with the bars stacked

(as in Fig. 1), giving to every ten bars six bottom- and four top-samples. The samples were taken diagonally across, as shown (points *a*, Fig. 1).

Assay-result, 403.5 ounces per ton.

(K) Sample taken from five bars center of top and five bars center of bottom, alternately (points *a*, Figs. 3 and 4).

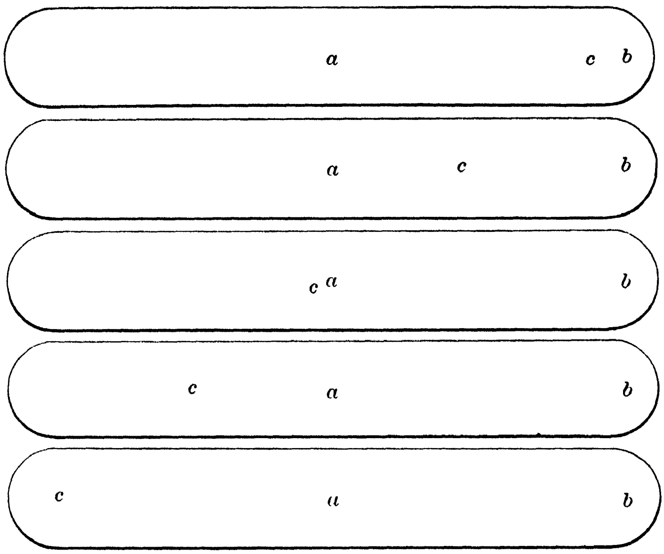
Assay-result, 403.2 ounces per ton.

(L) Sample taken from the lugs, five top and five bottom, alternately (points *b*, Figs. 3 and 4).

Assay-result, 407.5 ounces per ton.

(M) Sample taken diagonally across the top of the bars (points *c*, Fig. 3).

FIG. 3.



Assay-result, 396 ounces per ton.

(N) Sample taken diagonally across the bottom of the bars (points *c*, Fig. 4).

Assay-result, 408.9 ounces per ton.

The average of E and F is 402.5 ounces per ton, and represents the result obtained by the present method of sampling.

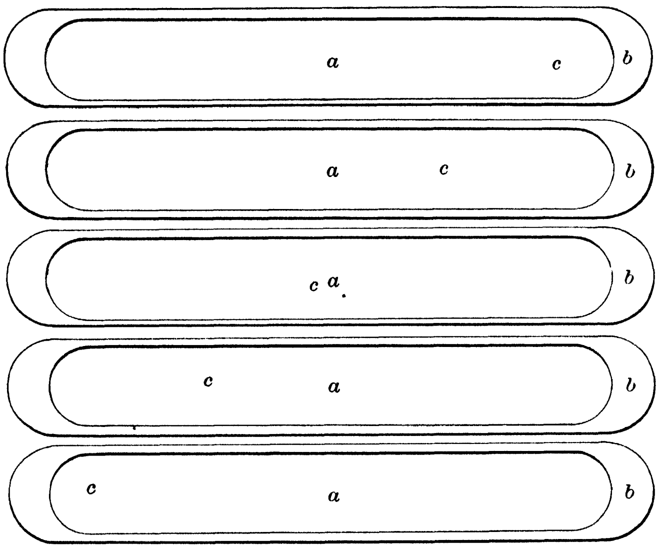
(O) Sample taken by sawing through the bars (see Fig. 5) and treating the saw-dust obtained like the corresponding sample (H) taken of bullion from the liquation-furnace.

The assay-result was:

No.	Ozs. per ton.
1,	410
2,	409.6
3,	409
4,	409.2
Average,	409.45

It will be seen that there is a considerable difference (in this case, 12.9 ounces per ton) between (M), taken from top, and (N), taken from the bottom of the bars; so that, in sampling six bars at bottom and four bars at top (as would be the case in taking samples from bars stacked, as shown in Fig. 1), the

FIG. 4.



advantage would be in favor of the seller. In this particular case, however, it amounts to only 1 ounce per ton.

What is most noticeable is the high and concordant results obtained by the two only really reliable samples, namely, (I) the dip- and (O) the saw-sample. They are for (I) 408.7 and for (O) 409.45 ounces per ton.

The results of the above experiments point to the fact that samples taken with the gouge (that is, chip-samples) from the top and bottom of the bars are inaccurate, and always too low. The best remedy in this case would be to run all the bullion obtained from the north furnaces through a liquation-furnace,

thereby providing a cleaner bullion, and also making it easy to obtain a dip-sample while the lead is being discharged into the moulds, as was the case in the experiments described in Section I., which prove the dip-sample to be, under such circumstances, perfectly accurate.

III.—Gouge- and Chip-Sampling.

Eight lots of bullion were sampled by the old method, the bars being stacked as in Fig. 1, and 6 samples being taken from the bottom and 4 from the top. They were then re-sampled by taking a chip out of the top and bottom of each bar.

The assay-results were as follows:

Lot.	Old method. Ozs per ton.	Chips from top and bottom. Ozs. per ton.
C 88,	441	439
" 89,	428	432
" 90,	392	384
" 91,	394	392
" 92,	376	374
" 93,	398	385
" 94,	492	489
" 95,	462	459

In all these lots except No. 89 the new method gives lower results than the old. Except as to Nos. 90 and 93, the difference amounts to from 2 to 3 ounces. This, in consideration of the further fact that both the above methods give results lower than either of the only accurate samples, namely, the dip- and the saw-samples, shows a great loss to the producer, and a corresponding gain to the refiner, by the adoption of chip-samples.

IV.—Distribution of Silver in the Bar.

It will be noticed that, for both lots of bullion described in Sections I. and II., the results obtained by the dip- and saw-samples are higher than those obtained by the gouge- or chip-sample, in sampling the bars (at points *c*, in Figs. 3 and 4 combined). This can only be explained on the assumption that the silver contents of the bullion vary in different parts of the bar, and that the richer parts escape sampling by the gouge- or chip-method, but are fairly represented in both the dip- and saw-samples. In order to collect data on this point, the following samples were taken:

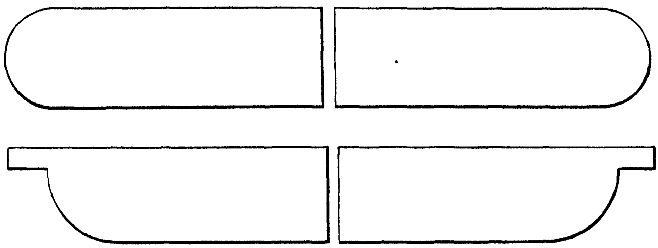
Lot I., from the liquation-furnace, was sampled by taking a gouge from the very center of the bar along the sawn surface.

Assay-result, 462 ounces per ton.

Lot II., north furnace bullion, was similarly sampled.

Assay-result, 400.3 ounces per ton.

FIG. 5.



In both these cases the results were lower than those of the dip- and saw-samples, which were 469.4 and 409.45 ounces respectively. The richer part of the bar, therefore, is not touched by the gouge-sample, but is represented in a dip- or saw-sample.

A section three-quarters of an inch thick was cut out of the center of two bars and laid off into nine portions, as shown in Figs. 6 and 7. Each portion was assayed separately, and the results will be seen in the two figures. Fig. 6 represents

FIG. 6.

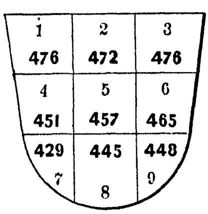
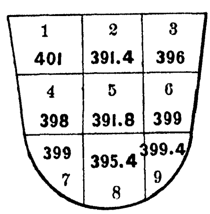


FIG. 7.

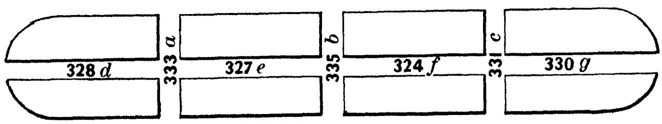


a bar from the liquation-furnace. In this case the results are very irregular; but as this liquation-bullion is all sampled by the dip-method, this irregularity is not commercially important. Fig. 7 represents a bar from the north furnaces. It will be seen that the outside portions of the bar, numbered 1, 4, 7, 3, 6 and 9, are by far the richest. In a gouge- or chip-sample practically none of this rich material is

taken. Some of the bullion in parts 1 and 3 is obtained, but only from the very top of the bar, which, as has already been shown, is poorer than the average of the bar (see M, Section II.); but in a saw- or dip-sample a fair proportion of all these parts is obtained (see Fig. 5).

A bar was taken from the liquation-furnace and sawn through transversely in three places; then each piece was sawn longitudinally at right angles to the upper face (see Fig. 8). The assay-results corresponding to the different saw-cuts are

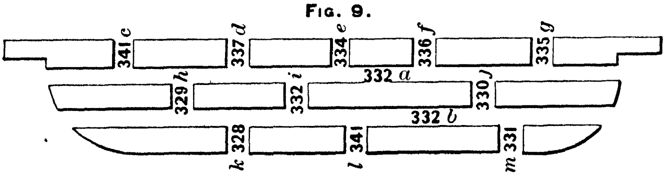
FIG. 8.



indicated on the figure. The assays were made on the saw-dust obtained from each cut. The dip-assay of this bar gave 334 ounces per ton.

Another bar was taken, and sawn as shown in Fig. 9. Two cuts were made longitudinally across the bar parallel to its upper face, and the three strips thus made were sawn through transversely, as shown in the figure. The results of assays are marked on the figure in the different saw-cuts. Dip-assay of bar, 336 ounces.

FIG. 9.



No definite data can be obtained from these results as to the composition of the interior of the bar in various parts, but they are interesting as showing that the silver-contents are very unevenly distributed through it.

Two bars were taken from the north furnaces and treated as explained in the preceding paragraph. The results of assays are marked in the different saw-cuts shown in Figs. 10 and 11. In Fig. 10 the middle cross-section shows a higher result than any of the others, while in Fig. 11 the three middle cuts exhibit about the average of the other cuts, containing the lowest

assay at *e*, the highest at *l*, and an average at *i*. It will be noticed in Fig. 11 that the samples contained more of the outside of the bar, which lies against the mould, and that, with the exception of the top-center saw-cut, the samples agree much more closely than is the case in Fig. 10.

Samples were taken from the outside (*i.e.*, that portion of the bar lying against the mould) of the bars from the liquation-furnace. Nos. 1, 2, 3, 4, 5, 6 and 7 were taken from the sides of the bar shown in Fig. 8, and Nos. 8, 9, 10, 11, 12, 13 and 14 were taken from the sides of the bar shown in Fig. 9. Samples 4, 5, 6 and 7 were taken just under the upper edge of the top of the bar, while the rest were taken about two-thirds down the side of the bar. The following are the assay-results:

Samples from Fig. 8.

No.	Ozs per ton.
1,	343
2,	335
3,	341
4,	343
5,	337
6,	343
7,	334
Average,	339.4

The dip-assay of this bar was 334 ounces per ton, showing that the outside, in contact with the mould, is the richest part of the bullion. None of this is obtained in the gouge- or chip-sample.

Samples from the Sides of Fig. 9.

No	Ozs. per ton.
8,	337
9,	337
10,	343
11,	335
12,	342
13,	335
14,	345
Average,	339.14

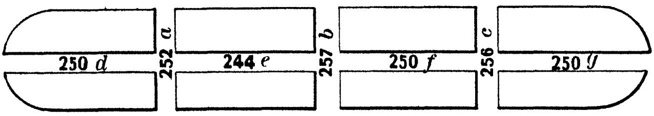
The dip-assay of this bar was 336 ounces per ton. These results confirm the statement made above.

The two bars from the north furnaces, shown in Figs. 10 and 11, were sampled from the outside. Only four samples were taken. The assay-results were:

No.	Ozs. per ton.
1,	257
2,	257
3,	257
4,	258

On comparing these assays with the others shown on Figs. 10 and 11, it will be seen that in all cases they are as high as, or higher than, the results obtained in any other part of the bar.

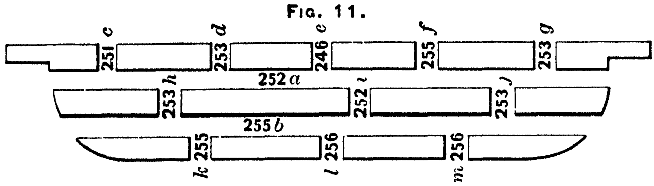
FIG. 10.



In order to gain further evidence as to the richness of the outside of the bars, samples were taken from the bullion, experiments upon which are reported in Sections I. and II. of this paper. The lots were sampled by taking a chip out of the side of the bar, the chips being distributed equally along the sides from top to bottom; *i.e.*, every five bars were sampled as shown by points *a*, Fig. 2.

Lot I. assayed 475 ounces per ton, against 469.4 ounces by dip- and 470 ounces by saw-sample.

FIG. 11.



Lot II. assayed 412 ounces per ton, against 408.7 ounces by dip- and 409.4 ounces by saw-sample.

These series of tests seem to establish the fact that there is no regularity in the distribution of the silver inside the bar, but that that part of the bar which lies directly against the sides of the mould is the richest, as already observed. Nothing from this part of the bar is obtained by the gouge-method of sampling the bars on top and bottom, in use up to the time when these experiments were made; but this portion is represented in a dip- or saw-sample. Hence, these two classes of

samples give higher and far more accurate results than chip-samples.

Several additional tests were made to prove further the accuracy of the dip- and saw-samples, and in all cases the two agreed to within half an ounce per ton. Other tests, taken from the sides and lugs of the bars, always assayed higher than the correct value of the bar.

V.—Loss of Lead by Volatilization.

The question of the loss of lead by volatilization in pouring or melting, and the consequent enrichment of the sample in silver, has been thoroughly studied by the writer. Samples of bullion from the liquation-furnace were weighed before and after melting, with the following results:

Sample. No.	Loss. Per cent.
1,	0.08
2,	0.06
3,	0.05
4,	0.07
5,	0.05

This loss, however, was not due to volatilization, but was caused by small particles of lead adhering to the sides of the crucible and sticking to the rod used for stirring. Three samples of bullion from the same lot were tried, as follows: 1500 grains were carefully weighed, placed in a red-hot crucible, and melted without stirring, a quick circular movement being given to the crucible instead. The lead was then poured into a small mould, and when cold was reweighed. The difference in weight was *nil* in two of the three cases, and the third gave a loss of half a grain, which would be 0.03 per cent. These tests prove conclusively that with care there need be no loss in melting soft bullion. As no oil is used in sampling by the gouge, treatment with benzine was omitted, except in the case of the saw-samples, which were greasy, owing to the saw having been well lubricated during the process. These were, therefore, all treated with benzine before melting down, and in this process showed a loss of 0.25 per cent., due to oil and greasy matter.

Of the bullion from the north furnaces, which is tapped direct from the furnace into a lead-kettle, then skimmed and ladled

into moulds, two samples were melted, as above. The loss was 0.166 per cent., caused principally by a little drossy matter, which forms and adheres to the crucible; also a few globules of lead, which sometimes stick to the crucible. In the case of the bullion from the liquation-furnace no dross forms, the lead having been considerably purified in passing through this furnace. The writer is of opinion that there should be no loss in melting soft bullion if care is exercised and the bullion is not allowed to remain in the crucible after it becomes molten. There is no loss by volatilization when bullion is poured as soon as melted.

In connection with lead-bullion assays, the writer always melts in a red-hot crucible, which is preferable to the ordinary method of putting the crucible and lead into the furnace to melt. It has the advantage of quickness and greater command over the work; and, by several tests which he has made in melting the ordinary bullion-samples, the writer has found it an admirable way of melting lead-bullion. The samples seem to be more homogeneous when melted without placing in a hot fire; the checks have come out very close; and where, previously, five or six checks had to be made, only one or two were found necessary after adopting the hot-crucible method. Besides, the danger of loss is considerably reduced. The crucibles are kept hot in a furnace alongside the operator.

There is a slight loss in hammering the samples, due to small pieces of lead breaking off from the edges; but this makes no difference in the result of the assay, as the pieces are of similar composition to the sample assayed.

These tests prove conclusively that, with careful work, no lead is volatilized, and consequently no enrichment in silver contents of bullion takes place, in melting the bullion-samples.

The loss in cupellation is not great with this bullion. Experiment showed a loss of 0.015 per cent., or, on 400-ounce bullion, 0.06 ounce per ton. The test was made by making a sample of similar composition to the bullion-sample and cupelling it with the bullion-sample in the same muffle-furnace. The loss in this instance was not of much importance; but, as a general rule, the operator needs to be very careful in cupelling bullion-samples, as the loss may otherwise be great.

VI.—SUMMARY.

Samples taken by chipping out of the top and bottom of each bar almost always give a result lower than the actual value of the bullion. This is due to the fact that that part of the bullion lying against the sides of the mould is the richest in silver, and that none of it is obtained in the chip-sample. If chip-samples are to be taken, it would be fairer to take chips from the top, bottom and side of each bar, these to be taken diagonally across the five bars when stacking. The tests reported in this paper prove that the only really accurate samples are the dip-sample and the saw-sample.

List of Assays Marked on Figures.

FIG. 6.		FIG. 7.	FIG. 8.		FIG. 9.	FIG. 10.	FIG. 11.
No.	Ozs Per Ton.	Ozs. Per Ton.	No.	Ozs. Per Ton.	Ozs. Per Ton.	Ozs. Per Ton.	Ozs. Per Ton.
1...	476	401	a....	333	332	252	252
2....	472	391.4	b ...	335	332	257	255
3. ..	476	396	c ..	331	341	256	251
4. ..	451	398	d.....	328	337	250	253
5....	457	391.8	e.. ..	327	334	244	246
6. ..	465	399	f	324	336	250	255
7.	429	399	g . ..	330	335	250	253
8 .	445	395.4	h ..		329		253
9. ..	448	399.4	i ...		332		252
			j		330		253
			k . ..		328		255
			l ...		341		256
			m. .		331		256

The Influence of Bismuth on Brass, and its Relation to Fire-Cracks.

BY ERWIN S. SPERRY, BRIDGEPORT, CONN.

(Buffalo Meeting, October, 1898.)

IT is a tradition in the brass industry that bismuth is an injurious element in brass, even more deleterious than antimony; but such a belief has lacked verification. The occasional presence of bismuth in commercial copper (although less frequent than was formerly supposed) led the author to investigate this tradition.

The following experiments were conducted in the same man-

ner as those described in my paper* on "The Influence of Antimony on the Cold-Shortness of Brass," to which the reader is referred for details. The purest Lake Superior copper was used, and after it had been melted the bismuth was introduced as an alloy of copper and bismuth. Zinc in the form of pure refined metal was next added, and the mixture was stirred and poured into an iron mould $\frac{5}{8}$ by $2\frac{5}{8}$ by 24 inches. All rolling was performed cold. For reasons previously mentioned,† a base alloy was used consisting of copper 60 per cent. and zinc 40 per cent.

Experiment No. 1.

Copper, 59.50; zinc, 40; bismuth, 0.50 per cent. Melted 5.5 pounds of copper, added 0.5 pound of an alloy of copper 90 and bismuth 10 per cent., and then 4 pounds of zinc. Rolled from 0.595 to 0.472 inch, a reduction of 20 per cent. Cracked badly on one edge, but otherwise the surface appeared unchanged. Annealed, and the plate fire-cracked on both sides. While many of the cracks were merely superficial, others penetrated the metal so deeply that further rolling became impossible. The fire-cracked plate is reproduced in Fig. 1. While this alloy is cold-short, it is less so than the corresponding‡ antimony alloy. The fracture is finely crystalline, as shown in Fig. 2.

Experiment No. 2.

Copper, 59.69; zinc, 40.06; bismuth, 0.25 per cent. Melted 5 pounds 6 ounces of copper, added 3.75 ounces of an alloy of copper 90 and bismuth 10 per cent., and then 3.75 pounds of zinc. Rolled from 0.655 to 0.495 inch, a reduction of 24 per cent. Cracked slightly on the edges, and a few checks appeared on the surface. Annealed, and fire-cracks formed on both sides of the plate. These cracks were quite numerous, but did not penetrate the metal as deeply as those in Experiment No. 1. Rolled from 0.495 to 0.416 inch, a reduction of 15 per cent., and the fire-cracks became greatly enlarged; many others also appeared. The latter were apparently *latent* fire-cracks, as the plate did not show any additional cracks on the edges. Such latent fire-cracks were noticed in succeeding experiments, and are evidently due to the presence of bismuth, as

* Page 176 of the present volume. † *Ibid.* ‡ *Ibid.*

the reduction in rolling was not sufficient to cause them. Rolled still further to 0.355 inch, a reduction of 29 per cent., and the plate cracked to pieces. The edge-cracks first formed did not increase in size perceptibly.

The following test was made on a rod* cut from an ingot which was cast at the same time as the plate. The results show that the cause of the cracks was not insufficient elongation.

Length, 12 inches; diameter, 0.270 inch; sectional area, 0.0572 square inch; breaking strain, 2560 pounds, or 44,700 pounds per square inch; elongation in 1 inch, 40 per cent.; in 6 inches, 27 per cent.; diameter of fracture, 0.205 inch; reduction of area, 42.3 per cent.

The cold-fracture of this alloy is finely crystalline, and the appearance so similar to that of the alloy containing 0.50 per cent. of bismuth that it is deemed unnecessary to show a reproduction.

Experiment No. 3.

Copper, 60.01; zinc, 39.90; bismuth, 0.09 per cent. Melted 5 pounds of copper, added 1⅓ ounces of an alloy of copper 90 and bismuth 10 per cent., and then 3 pounds and 6 ounces of zinc. Rolled from 0.660 to 0.485 inch, a reduction of 26 per cent. No cracks appeared during this reduction. Annealed, and the surface of the plate presented an unchanged appearance; no traces of fire-cracks were visible. Rolled to 0.420 inch, a reduction of 13 per cent., and an enormous number of fire-cracks appeared on both sides of the plate. No edge-cracks formed during this reduction. The cracks which were produced during the latter reduction were evidently of a latent nature, like those in Experiment No. 2. They penetrated the plate so deeply that further rolling was impossible. The cold-fracture of this alloy is finely crystalline, and is reproduced in Fig. 3.

Experiment No. 4.

Copper, 59.95; zinc, 40; bismuth, 0.05 per cent. Melted 5.5 pounds of copper, added 0.5 pound of an alloy of copper 99 and bismuth 1 per cent., and then 4 pounds of zinc. Rolled from 0.594 to 0.420 inch, a reduction of 29 per cent. No

* Not annealed after being turned.

cracks appeared. Annealed, and rolled to 0.319 inch, a reduction of 24 per cent. A few slight edge-cracks formed during this reduction, but otherwise the sheet remained unchanged. Annealed, and fire-cracks appeared on both sides of the sheet. Many of the cracks were so deep that the plate could not be rolled again. Rolled another portion from 0.420 to 0.371 inch, a reduction of 11 per cent., and then annealed. No fire-cracks could be seen. Rolled to 0.325 inch, a reduction of 12 per cent., and *latent* fire-cracks appeared in large numbers. The position of these cracks was different from those on the other plates; many were transverse, others diagonal, and others longitudinal. Rolled without annealing to 0.189 inch, a reduction of 41 per cent. No edge-cracks formed, and the fire-cracks seemed to have become obliterated, which gave the sheet* the appearance of perfect homogeneity. Rolled the sheet still further to 0.058 inch, a total reduction of 84 per cent., and it cracked so badly on the edges that additional reduction was almost impossible.

By an examination of the fracture of this alloy, shown in Fig. 4, one can readily see that crystallization has nearly disappeared.

Experiment No. 5.

Copper, 59.98; zinc, 40; bismuth, 0.02 per cent. Melted 5 pounds and 12.75 ounces of copper, added 3.25 ounces of an alloy of copper 99 and bismuth 1 per cent., and then 4 pounds of zinc. Rolled from 0.590 to 0.504 inch, a reduction of 14 per cent. Did not crack on the edges. Annealed, and rolled to 0.384 inch, a reduction of 23 per cent. Neither edge-cracks nor latent fire-cracks appeared. Annealed, and rolled to 0.052 inch, a reduction of 86 per cent. At the thickness of 0.125 inch slight cracks began to appear on the edges, and at 0.052 the reduction had reached the maximum amount. No fire-cracks were detected during the rolling-process. This alloy seemed to roll nearly as well as brass free from bismuth; therefore, 0.02 per cent. of bismuth represents the dividing-line between satisfactory and unsatisfactory brass.

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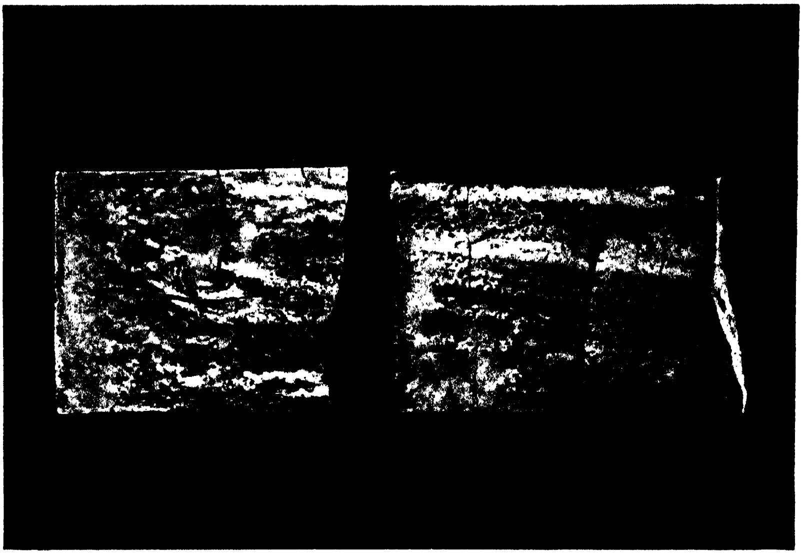
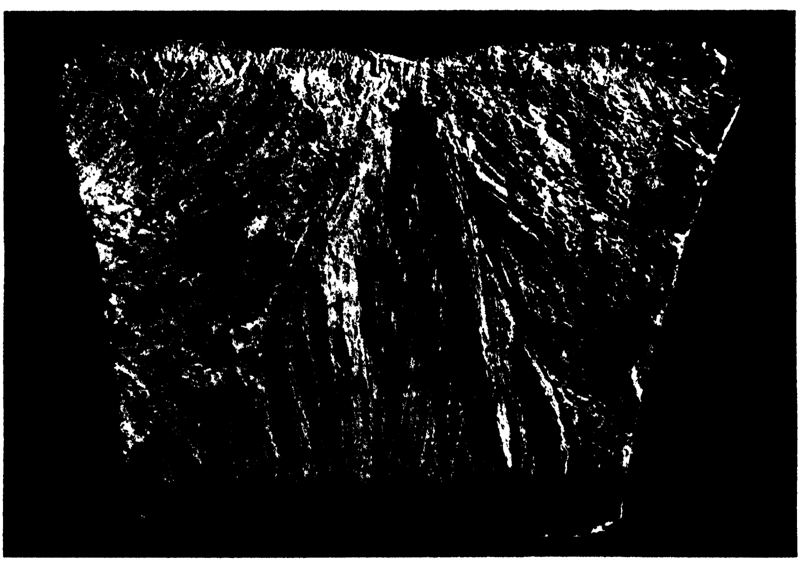
* When such sheet is rolled, cracks of this nature either close so as to become invisible to the naked eye or form slivers, by the edges of the cracks overlapping one another and then becoming rolled down. The invisible cracks do not weld together; and while, under ordinary conditions, they are difficult to detect, they are readily made apparent by polishing the sheet.

FIG. 1.



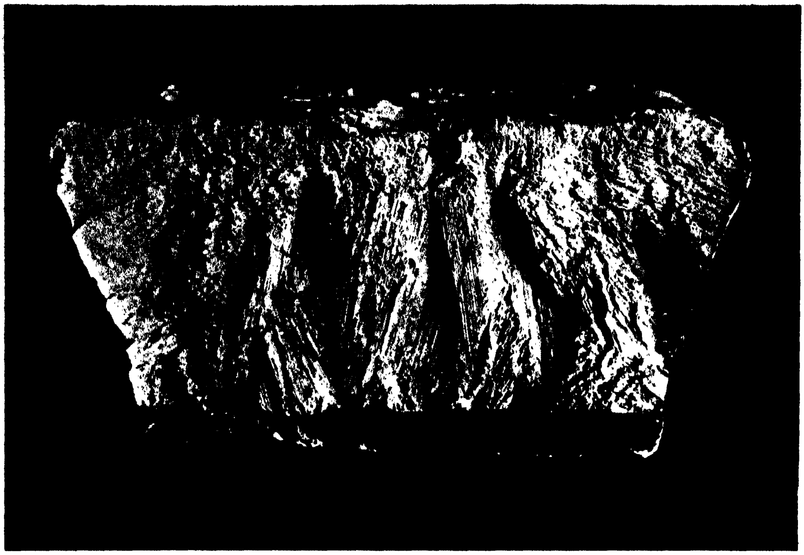
BRASS CONTAINING 0.50 PER CENT OF BISMUTH.
WHICH FIRE-CRACKED DURING ANNEALING.

FIG. 2.



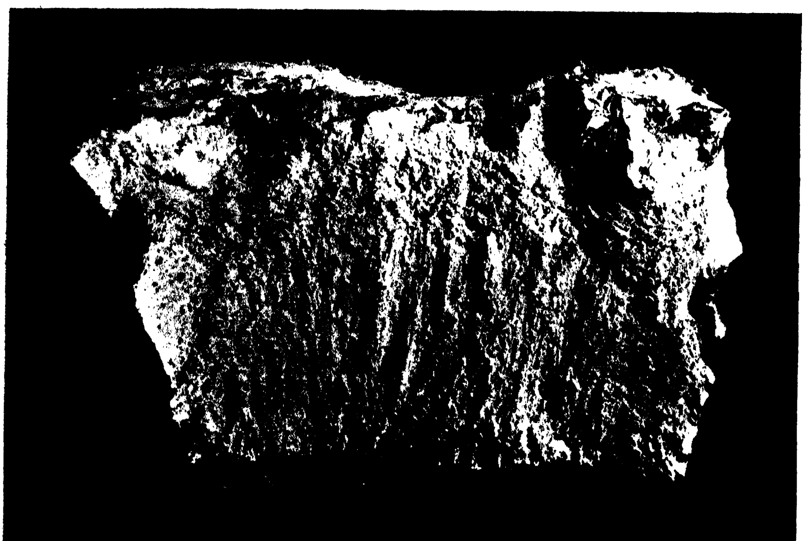
BRASS CONTAINING 0.50 PER CENT OF BISMUTH.
COLD FRACTURE, MAGNIFIED TWO DIAMETERS.

FIG. 3.



BRASS CONTAINING 0.09 PER CENT OF BISMUTH.
COLD FRACTURE, MAGNIFIED TWO DIAMETERS.

FIG. 4.



BRASS CONTAINING 0.05 PER CENT OF BISMUTH.
COLD FRACTURE, MAGNIFIED TWO DIAMETERS.

FIG. 5.

BRASS CONTAINING 0.02 PER CENT OF BISMUTH.
COLD FRACTURE, MAGNIFIED TWO DIAMETERS.

The following tests were made on sheet rolled from 0.384 to 0.053 inch, a reduction of 86 per cent.:

	As rolled.	Annealed.
Size, ...	0.482 by 0.0535 in.	0.475 by 0.0535 in.
Length, ...	12 in.	12 in.
Sect. area, . . .	0.0257 sq. in.	0.0254 sq. in.
Breaking-strain, . .	2816 lbs.	1510 lbs.
Breaking-strain per sq. in.,	109,000 lbs.	59,000 lbs
Elong. in 1 in., . . .	4 per cent.	47 per cent.
Elong. in 8 in., . . .	1 per cent.	37 per cent.
Size of fracture, . .	0.474 by 0.0515 in.	0.365 by 0.039 in.
Reduction of area, . .	5 per cent.	44 per cent.

The fracture of this alloy shows no traces of crystallization, and compares favorably with that of brass free from bismuth or other injurious elements. The comparison of the results obtained by rolling this alloy, with those obtained on pure brass of the same composition, show that practically the same reductions can be given. The fracture is reproduced in Fig. 5.

Conclusions.

While it has been known for some time that bismuth is an injurious impurity in brass, the author believes it has not been demonstrated before that this element is a cause of fire-cracks. The reason for this phenomenon is not well understood, but there is evidently a segregation of bismuth or an alloy of bismuth which is devoid of sufficient strength to stand the necessary strain.

The results obtained by Stead, in his researches* on the microscopic examination of alloys, throw some light on the subject. By the examination of an alloy of copper 99 and bismuth 1 per cent., he found that a segregation takes place, and that if such an alloy is bent, fracture will take place along the lines of this segregation. It is reasonable to assume that the action of bismuth on brass is similar.

It must not be supposed that all fire-cracks in brass are due to bismuth. As a matter of fact, it is very doubtful whether this element is usually the cause of such trouble; in the majority of instances the amount of bismuth in copper is not large enough to interfere with its employment in the manufacture of brass.

* *Jour. Soc. Chem. Industry*, vol. xvi., 1897, p. 208.

The behavior of these alloys when forged hot is worthy of notice. The alloy containing 0.50 per cent. of bismuth could not be forged at any heat. The 0.25 per cent. alloy behaved in a similar manner. The alloy containing 0.09 per cent. of bismuth would forge to a thin edge, but if bent upon itself fractured at the bend. The 0.05 per cent. alloy likewise fractured at the bend. The 0.02 per cent. alloy would forge to a thin edge and bend over on itself, showing but few cracks, but did not behave as well in this respect as brass containing no bismuth.

The results obtained in the preceding experiments are collated in the following summary:

1. Bismuth renders brass cold-short, and is similar in this respect to antimony, although the effect is not as marked.
2. Bismuth is a cause of fire-cracks in brass.
3. High brass intended for cold-rolling should not contain over 0.01 per cent. of bismuth.
4. Bismuth produces hot-shortness in brass.
5. Bismuth is a cause of *latent* fire-cracks in brass.

In the use of the term *latent* in connection with fire-cracks it is the author's belief that while annealing is the cause of their existence, they do not exist as cracks in the metal after annealing, but as lines of inferior cohesion in an apparently homogeneous mass; rolling, however, develops them, and to all appearances they then partake of every characteristic of true fire-cracks. It seems advisable, therefore, to apply such a distinguishing term, especially as *latent* fire-cracks are liable to be mistaken for *rolling*-cracks.

A Modification of Bischof's Method for Determining the Fusibility of Clays, as Applied to Non-Refractory Clays, and the Resistance of Fire-Clays to Fluxes.

BY H. O. HOFMAN, INSTITUTE OF TECHNOLOGY, BOSTON, MASS.

(Buffalo Meeting, October, 1898.)

In determining experimentally the fusibility of clays, two kinds of methods may be distinguished—the direct and the indirect. Of the direct methods, that of Seger has found

much favor. It consists in placing in a crucible the clay to be tested with Seger cones* (graded mixtures, the melting-points of which are known), and heating in a suitable furnace until a cone is found which shows the same behavior in the fire as the clay. Different apparatus is used for refractory and non-refractory clays, the standard for a refractory or fire-clay being that it shall not melt before Seger cone No. 26, or at an approximate temperature of 1650° C. The details of Seger's method of comparing fire-clays with his standard refractory cones Nos. 26–36 have been given in a previous paper.† For determining the fusibility of non-refractory clays, a special form of gas-furnace‡ is required, in which, with a good pressure of gas, Seger cone No. 25, the highest of the non-refractory mixtures, can be melted down in from 4½ to 5 hours. It may be said, however, that the melting-down of cones Nos. 20–25 is often difficult.

Of the indirect methods, that of Bischof§ has been extensively used. It consists in toning-up weighed samples of the clay to be tested with increasing quantities of an intimate mixture of equal parts of chemically pure silica and alumina, and forming them into small prisms, to be heated with a prism of Saarau fire-clay (equal to Seger cone No. 36) to above the melting-point of wrought-iron. The sample which shows the same behavior in the fire as the Saarau prism is the critical mixture, and the amount of toning-up substance needed forms the criterion of the fusibility of the clay.

Beyond question, the direct method is the simpler, since the clay need only be ground, moulded, dried and heated with standards which are always uniform and can be bought in the market at a low price. The first test is made with Seger cone No. 26, to see if the clay is refractory or not, and the work is then continued with the Deville or the Seger gas-furnace. There is no drying and igniting, no weighing-out of clay and of fluxes, and no intimate mixing—all of which are tedious operations. If many tests have to be made, the direct method

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* *Thonindustrie-Zeitung*, 1893, p. 1252; *Berg- und Hüttenm. Zeitung*, 1894, p. 119; *The Clay Worker*, August, 1897.

† *Trans.*, xxv., p. 3.

‡ *Thonindustrie-Zeitung*, 1896, No. 63; *Berg- und Hüttenm. Zeitung*, 1897, p. 21.

§ *Dingler's Polyt. Jour.*, 1870, vol. cxcvi., pp. 438, 525; cxcviii., p. 396.

will always be followed. The indirect method, however, has the advantage that only one standard and one furnace are required. Considering the difficulties often encountered in melting down in the gas-furnace clays that are near the refractory line, and the advantage in this respect of using only one furnace (the one for refractory clays, which is cheap, and which anyone can build for himself), and simply determining how much refractory material is necessary to bring a non-refractory clay up to the required standard—we must confess that there are conditions under which the indirect method can hold its own.

The work to be described in this paper formed part of a thesis of Messrs. J. L. Newell and G. A. Rockwell, of the Class of '95, who with much care carried out the large number of tests required to verify the method.

In these experiments the Bischof standard (Saarau clay, corresponding to Seger cone No. 36) was changed to Seger cone No. 26, which, as previously observed, forms the line of separation between refractory and non-refractory clays, the non-refractory clays being toned up until they showed the same behavior in the fire as Seger cone No. 26. This was done because it was of more interest to find out how far the non-refractory clay stood below the point of being a fire-clay than how much refractory material would have to be added to bring it up to the Saarau or Seger cone No. 36 standard. Moreover, it was thus possible to work at a lower temperature, with a saving in time and gas-carbon, and a prolongation of the life of the furnace-lining.

The method may of course be varied. For example, the writer has toned up low-grade fire-clays with bauxite until they showed the same behavior as certain high-grade fire-clays, or their equivalents in terms of Seger cones.

The silica used in the experiments was quartz, ground to pass a 100-mesh sieve and purified by boiling with nitrohydrochloric acid. Upon analysis, it showed 99.88 per cent. of SiO_2, and was assumed to be pure. The alumina was obtained from the Solway Process Company, Syracuse, N. Y. An analysis furnished by the makers showed Al_2O_3, 98.46; SiO_2, 0.25; Na_2O, 0.50; Fe_2O_3, 0.04; loss by ignition, 0.75 per cent. As the substance readily absorbs moisture, a sample was ignited, which gave a loss of 6.42 per cent.; and an allowance for this loss was made

in all the work. The clays tested were kindly furnished by Prof. Edward Orton, Jr. Their composition and the results of the tests are shown in the accompanying table.

Analyses of Clays and Results of Tests.

Sample No.	26*.	25*.	8*.	22*.	24*.	23*.	1982†.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent	Per cent.	Per cent.
SiO_2	64.10	55.60	57.10	57.45	57.15	49.30	43.94
Al_2O_3	21.79	24.34	21.29	21.06	20.26	24.00	11.17
H_2O comb.	6.05	6.75	6.00	5.90	5.50	9.40	3.90
Total	91.94	86.69	84.39	84.41	82.91	82.70	59.01
F_2O_3	2.51	6.11	7.31	7.54	7.54	8.40	3.81
CaO	0.10	0.43	0.29	0.29	0.90	0.56	11.64
MgO	0.58	0.77	1.53	1.22	1.62	1.60	4.17
K_2O	2.62	3.00	3.44	3.27	3.05	3.91	2.90
Na_2O	0.03	0.09	0.61	0.39	0.58	0.17	0.71
Total	5.84	10.40	13.18	12.71	13.69	14.64	23.23
Moisture	1.10	2.65	1.30	1.90	2.70	1.20	15.66‡
Grand total	98 88	99.74	98.87	99.02	99.30	98 54	98.00§
Stiffening ingredient, p.c.	20	40	60	80	80	100	180

The method of operation was to weigh out samples of 1 gramme of the clay to be tested (the moisture being allowed for); mix them severally with 0.1, 0.2, 0.3, etc., gramme of the silica-alumina flux, in small porcelain dishes; turn out each mixture upon a glass plate; moisten it with a 10 per cent. dextrine solution; work it with a spatula until it had acquired the right consistency, and mould it into the form of the small-size Seger cone.|| When dried, three of them were placed in a crucible with a Seger cone, No. 26, and so heated in the Deville furnace as to melt down the Seger cone. In addition to the usual 30 grammes of paper and 200 grammes of charcoal, from 920 to 925 grammes of gas-carbon were required, with a pres-

* Analyzed by N. W. Lord.

† Analyzed by E. Orton, Jr.

‡ Includes CO_2.

§ Includes P_2O_5, 0.10 per cent.

|| *Trans.*, xxv., pp. 10 and 11.

sure of blast of about 1 inch of water. A fusion required about 35 minutes.

In the table above, the clays are arranged according to their degree of refractoriness. Sample No. 26 requires 20 per cent. of flux to raise its melting-point to that of Seger cone No. 26; sample No. 25 requires 40 per cent., and so on.

In order to check the method, tests were made in two ways with large-size Seger cones. In the first, cones Nos. 1 to 25 were pulverized and toned up with the silica-alumina flux. It was found that 0.036 gramme of the flux, added to 1 gramme of Seger-cone substance, would raise its melting-point to that of the next higher number. In the second, two clay-samples, Nos. 22 and 23, which, according to indirect preliminary tests, ought to melt down at the same time as Seger cone No. 4, were placed in a graphite crucible, the bottom of which had been tamped with refractory clay, and were heated in a coke-furnace with under-grate blast. In two hours the test was finished. The two clays showed in the fire approximately the same behavior as the Seger cone, thus again proving the accuracy of the method.

This modification of Bischof's indirect method may also be used for determining the resistance of fire-clays or fire-bricks to the corroding influence of sodium chloride, sodium sulphate, sodium carbonate, potassium carbonate, etc., to which they are exposed in glass-pots, or to calcium carbonate, lead oxide, iron oxides, etc. To illustrate the operation, samples of 1.5 grammes of clay are mixed severally with 5, 10, 15, etc., milligrammes of flux, formed into small-size Seger cones, and heated in the Deville furnace with Seger cone No. 26, in such a way that the Seger cone will melt. Here, again, the sample which shows the same behavior as the Seger cone will be the critical mixture, and the percentage of flux it contains will form the criterion of the clay's resistance to corrosion. It is true that the results obtained do not altogether determine the suitability of a clay for the manufacture of glass pots, as the requirements* of such a clay are not only that it shall resist heat and fluxes, but that it shall be highly plastic and burn

* Seger-Cramer, *Thonindustrie-Zeitung,* 1897, p. 47.

dense at a comparatively low temperature. Nevertheless, the results obtained by the method form a valuable guide in the making up of mixtures. This was proved to the writer by the results with certain clays, selected on account of their good behavior under the tests, which turned out a true prediction of what occurred on a large scale afterwards.

The method can, of course, be applied to a number of cases where an acid furnace material is to be exposed to the corrosion of a basic charge, and *vice versa*.

Does the Size of Particles have any Influence in Determining the Resistance of Fire-Clays to Heat and to Fluxes?

BY H. O. HOFMAN AND B. STOUGHTON, INSTITUTE OF TECHNOLOGY, BOSTON, MASS.

(Buffalo Meeting, October, 1898.)

BEFORE examining a fire-clay in the laboratory for its resistance to heat or to fluxes, the sample is always ground to an impalpable powder. But when the clay is actually used for the manufacture of bricks, blocks, pots, etc., it is not ground to a uniform size, the particles varying from coarse grains to the finest slimes. The natural inference is that the tests with finely-ground substances will give lower results than if the materials are tested just as they are going to be used. The following experiments were made to find out how far this inference is justified. The method employed for fusion was Seger's direct method,* and, for fluxing, the modified Bischof method, described in the paper on that subject presented at the present meeting.† The first question to be decided was, how large the test-cones ought to be made, to include representative proportions of the different-sized particles composing the mixture. This was subjected to a screen-analysis with the following results :

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* *Trans.*, xxiv., p. 51; xxv., p. 3. † See page 435 of present volume.

TABLE I.—*Results of Screening.*

	Grammes.	Per Cent.
Before screening	74.05	......
On 8-mesh	16.1	22.1
On 12-mesh	16 6	22.8
On 16 mesh	6.9	9.5
On 20-mesh	6 6	9.0
On 30-mesh	7.4	10.1
On 40-mesh	2.5	3.4
On 60-mesh	4.0	5.5
Through 60-mesh	12.8	17.6
Lost in screening	1.15	......

After a number of preliminary trials it was decided to choose as a standard the large-size Seger cone,* $\frac{3}{4}$ inch at base and $2\frac{3}{8}$ inches in height. The results given in Table II. show that the choice was a correct one.

This table represents eight cones, made up from a sample of the mixture, which had been moistened with water containing a small amount of dextrine. Each cone, when dried, was first weighed, then disintegrated by rubbing in a Wedgwood mortar and sifted through the same screens with which the first sizing-test had been made; the resulting different-sized particles were then all separately weighed and their percentages calculated. The table shows the figures for 8-mesh and coarser sizes to be irregular. This was to be expected, as a few grains being a little larger or smaller than the average would cause a considerable difference in the percentage in view of the small weight of the cone. What went through an 8-mesh and remained on a 16-mesh screen showed more regularity; and the sizes from 16- to 60-mesh give very uniform data. That the material finer than 60-mesh should again show some differences was to be expected, as the fine clay-substance, adhering more or less to the coarser grains, could not be evenly separated by mere rubbing, which is all that can be used.

As the Seger cones for refractory clays, Nos. 26 to 36, are only $\frac{3}{8}$-inch at base and $\frac{25}{32}$-inch high, large-size refractory cones had to be specially made as standards for the tests. The Deville furnace, and the crucibles, with their lids and supports, were of the same general character as those described in a previous paper;† only, the dimensions were larger.

* *Trans.*, xxiv., p. 54.

† *Trans.*, xxv., p. 5.

TABLE II.—*Weights and Proportions of Different Sizes in a Large Seger Cone.*

Size.			Cone. No. 1. Weight. Gms.	No. 1. Per cent.	No. 2. Weight. Gms.	No. 2. Per cent.	No. 3. Weight. Gms.	No. 3. Per cent.	No. 4. Weight Gms.	No. 4. Per cent.	No. 5. Weight. Gms.	No. 5. Per cent.	No. 6. Weight. Gms.	No. 6. Per cent.	No. 7. Weight. Gms.	No. 7. Per cent.	No. 8. Weight. Gms.	No. 8. Per cent.
Before disintegrating......			7.950	......	7.670	......	6.833	......	8.715	.....	7.434	......	7.283	......	7.396	......	8.462	......
After disintegrating there	Remained on sieve.	8-mesh .	1.605	20.4	2 062	27.2	1.840	27.1	1.872	21.6	2.162	29 1	1.657	22.8	1.305	17.9	2.277	27.3
		12-mesh..	1.857	23.6	1.502	19.9	1.372	20.2	1.939	22.4	1.535	20.7	2.127	29.3	1.942	26.7	2.207	26.5
		16-mesh..	0.950	12.0	0.685	9 0	0.514	7.8	0 855	9.8	0.725	9.8	0.718	9.9	0.875	12.0	0.750	9.0
		20-mesh..	0.820	10.4	0.645	8 5	0.587	8.8	0.742	8.8	0 640	8.8	0.637	8 8	0.665	9.2	0.707	8.5
		30-mesh .	0 825	10.5	0 782	10.3	0.669	9.8	0 860	9.9	0.745	10.0	0.680	9 3	0.774	10.7	0.762	9.0
		40-mesh..	0.277	3.5	0.292	3.9	0.254	3.8	0.312	3.6	0.245	3.3	0.252	3.5	0.264	3 6	0.249	3.0
		60-mesh..	0.442	5.7	0 430	5 8	0.422	6.2	0.572	6 6	0.380	5.1	0.352	4.9	0.417	5.9	0.407	4.9
	Passed through sieve.	60-mesh	1.095	13.9	1.168	15.4	1.117	16 3	1.498	17.3	0.982	13.2	0.832	11.5	1.023	14.0	0 980	11.8

The furnace, of ⅛-inch sheet-iron, was 25 inches high and 12 inches in diameter; the cast-iron plate, 3¾ inches from the bottom, was 1 inch thick, had a central opening 2 inches in diameter surrounded by four rows of ¼-inch holes. The lining, of sintered magnesite from the Fayette Manufacturing Company, Pittsburgh, Pa., was 3¼ inches thick at the bottom and 2¾ inches at the top, making the inner dimensions of the furnace: diameter at bottom 5½ inches, at top 6½ inches, height 20 inches. The inner diameter of the air-inlet pipe was 1 inch. The outside dimensions of the crucibles were: diameter 2½ inches, height 3 inches, thickness of wall ¼ inch; the thickness of the lids ¼ inch; the size of the supports: diameter 2½ inches, height 2⅜ inches. The method of firing differed slightly from that pursued with the small furnace. With the latter the blast is slowly started, 30 grammes of paper are ignited and pressed down into the furnace, and then 200 grammes of charcoal are charged, to be followed by the required amount of gas-carbon. In the large furnace the space around the crucible-support was filled with small-size gas-carbon (about 40 grammes) before the paper (50 grammes) and charcoal (250 grammes) were introduced to kindle the increased amount of gas-carbon. Table III. gives the leading details about the manner of working with the furnace.

TABLE III.—*Operation of the Testing-Furnace.*

Seger Cone.		Fuel-Charge			Blast-Pressure, Inches Water.	Furnace, Initial Condition.	Time Required for Experiment. Min.
No.	Form After Fusion.	Paper, Grammes.	Charcoal. Grammes.	Gas Carbon, Grammes.			
26.........	Lenticular.	50	250	2900	2	Cold.	60
31.........	Globular.	"	"	3750	3	Hot.	50
32.........	"	"	"	4000	3	Cold.	55
33.........	"	"	"	2300	3½	Hot.	35
34.........	"	"	"	2400	3½	Hot.	40

The charcoal was passed through a small Blake crusher, set to 1½ inches, and then sifted through a 3-mesh screen, and the fines discarded. The gas-carbon was broken in the same way, and screened through a 1-mesh sieve. The under-size was passed through a 3-mesh screen to remove the fines. What

remained on the 1-mesh sieve was passed through the Gates laboratory-crusher, set to ⅜-inch, and was then used for filling the space around the crucible-support, or reserved for the small Deville furnace.

Table IV. gives the results of fusing- and fluxing-tests of mixtures just as they are to be used in making bricks, blocks, pots, etc., and of the same mixtures ground fine, as is the usual custom in laboratory tests. It shows only a slight difference in the results, and proves that, in testing fire-clay mixtures, the samples can be safely ground fine and compared with the small-size Seger cones in the ordinary Deville furnace. Exceptional conditions only would make it necessary to test the mixtures just as they are going to be used.

TABLE IV.—*Comparative Tests of the Same Material in Natural Condition and Finely Ground.*

Mixture.		Material as Received.		Same Material Finely Ground.	
Mark.	Tested for Resistance to.	Fusing-Test, Equal to Seger Cone, No.	Fluxing-Test, Grammes of $Ca\ CO_3$ to 1.5 of Mixture.	Fusing-Test, Equal to Seger Cone, No.	Fluxing-Test, Grammes of $Ca\ CO_3$ to 1.5 of Mixture.
A.....	Heat.	34	......	34	......
B.....	Heat and fluxes.	33	0.155	33	0.160
D.....	Heat.	34	......	34	......
E.....	Heat and fluxes.	34	0.180	34—33	0.180
H.....	Heat and fluxes.	33—34	0.160	33—34	0.160
No. 1	Heat and fluxes.	31	0.100	30	0.120

A New Assay for Mercury.

BY RICHARD E. CHISM, CITY OF MEXICO.

(Buffalo Meeting, October, 1898.)

THE dry methods of assaying mercury-ores and other combinations of mercury all rest upon the volatility of this metal as a beginning.

After the separation of the mercury in the form of vapor from the matrix upon which the assay is performed, the mercurial vapors are either condensed upon cold surfaces or received upon a gold plate into which the mercury is absorbed.

In the first case the product is recovered and weighed as metallic mercury in an isolated state.

The distillation-processes are practiced in either earthen or iron retorts, sometimes made of gas-pipes or in other provisional ways, or in glass tubes and upon quantities varying from a kilogramme or two down to a few grammes.

In any case the results of an ordinary distillation-assay are sure to be uncertain and unreliable, through loss of mercury, for close work. There is a great deal of difficulty in sealing all the joints of the apparatus, so as to prevent the escape of mercurial vapor. The metallic mercury after condensation is, at least in part, divided into minute globules, which are very elusive and frequently include particles of dust or other foreign bodies.

As a result the distillation-processes ought to be, and probably are, relegated to the class of "rough tests," in which capacity they are very useful. The sight of the metallic drops has often loosened the purse-strings of the doubting investor.

The production of mercury on a large commercial scale is well known to depend upon the utilization of ores of very low grade; and where this is practiced, it long ago became necessary to discard the distillation-methods of assay in favor of others that would give more exact results and admit of closer comparisons.

Such are the assays in which the mercury is determined in combination with gold; there being no simple wet assays that give practical results comparable with this class of dry assays.

The assays with gold are practiced upon quantities of ore ranging from 10 to 2 grammes; the richer the ore the less of it should be used. Never should there be so much mercury in the charge that the distillate hangs in drops upon the recipient.

The sample is mixed with iron, lime, litharge or other desulphurizing flux, and is placed in a porcelain crucible, which is then closed with a close-fitting concave cover, made of fine gold. The concavity of the cover is filled with distilled water, and the lower part of the crucible is then carefully heated. The mercury is volatilized, and deposits upon the gold cover as a white stain, like frosted silver.

When the operation is supposed to be ended, the cover is removed, carefully washed, dried in a desiccator, and then

weighed. At the beginning of the operation the cover has also been weighed, so that the increase of weight in the second weighing is due to the presence of mercury. Thus the percentage of this metal in the original charge of ore is easily ascertained.

This is, theoretically and practically, almost a perfect method. Repeated trials have shown agreements in duplicate assays to within 0.04 per cent. on poor ores, containing less than 0.5 per cent. of mercury, and within less than 0.5 per cent. on ores containing as high as 30 per cent. of mercury.

Only the very best analysts would be likely to get closer results with the wet methods at greater expenditure of time and money and no corresponding practical gain.

However, the gold-cover method has its drawbacks. The cover is expensive; scarcely less than an ounce of gold can be used if the cover is to stand many assays, for fine gold is quite soft, and the cover must be stiff enough, and have body enough, to stand handling without bending.

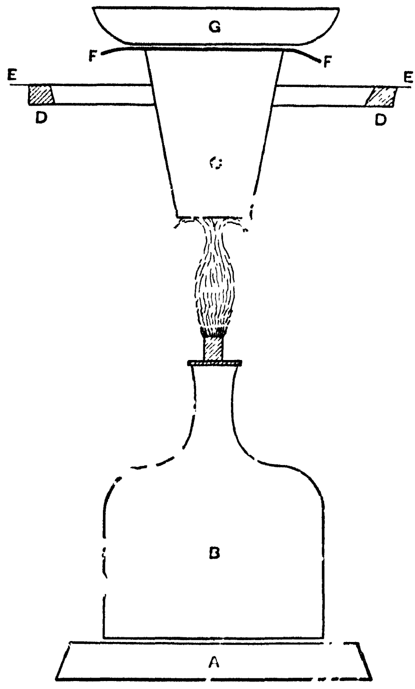
After the assay is finished, the mercury must be driven out of the cover by heat; and if the heating is not most carefully done, some of the gold will volatilize with the mercury. Notwithstanding all precautions, the gold cover, after a few determinations, begins to get spongy and afterwards to flake off. There is a considerable loss of weight; the assays are uncertain and discrepant in their results; and at last the cover must be remelted with new gold added, and then rerolled and rewrought.

Finally, the visible presence of gold is a temptation to irresponsible helpers and chance visitors. When goldsmiths are employed to form the covers, there is no security of receiving back all the gold with which they may have been furnished; or, if they themselves furnish the gold, their certificates, verbal and written, often cover more copper than the analyst knows of or desires.

There was, some time ago, a perfect mania in this republic for quicksilver-discoveries. My assay-office was often full of samples, generally barren, but all firmly believed to contain unbounded quantities of the liquid metal. Under these circumstances I early turned from leaky gas-pipe retorts and brittle glass tubes, and found a certain measure of relief in the gold-cover method of assay.

The drawbacks of the gold-cover are, however, strongly accentuated when one employs or comes into contact with persons whose ideas about portable property do not strongly distinguish between *meum et tuum*. Hence I sought for a method not so committed to the use of articles of high value, and thought first of using silver in the place of gold.

I have never seen any statement of the comparative affinity



Apparatus for Mercury-Assay, One-half Actual Size.

A Base of Retort Stand. B Spirit-Lamp. C Retort or Annealing-Cup. D D Retort-Stand Ring, which serves as support to the Apparatus. E E Tin Shield. F F Silver Foil for receiving the mercury. G Cooling-Cup.

of gold and silver for mercury, but possibly there is little difference between the metals in this respect, and perhaps silver has the stronger attraction of the two.

I desired also to get rid of the necessity of driving out the mercury by heat from the apparatus after every assay, and, lastly, I wanted to make a durable and cheap apparatus. The result of my experiments, as tested by considerable practice, is shown in the accompanying figure.

The Recipient.—The substance which I use to receive the vaporized mercury is, as I have before indicated, pure silver, and I use it in the form of foil, such as is commonly sold by dealers in assayers' supplies.

By actual measurement with a micrometer, the foil which I have been using is 0.02 millimeter in thickness. It costs in the United States about 5 cents gold per gramme.

At each assay I use a piece about 5 centimeters square, weighing about 0.6 gramme, so that the actual cost for each assay is about 3 cents, if the foil is not used over again. But it can be used, with very ordinary care, several (at least three) times, so that the cost becomes less than one cent per assay.

The Refrigerator.—For the purpose of cooling the silver-foil I make use of a silver dish of a wide pattern like an evaporating-dish. This form was adopted precisely for the reason that it facilitates the cooling for which it is designed. I chose silver because it has the highest heat-conducting power of all the metals, but copper approaches it so very nearly in that respect that a copper dish would practically be just as good, while it would cost less. The silver used in this dish is coin-alloy about 900 fine.

The dimensions are: 5.5 centimeters diameter at bottom, 6.5 centimeters at the top, and height somewhat over 1 centimeter.

It holds, comfortably filled, a little more than 20 cubic centimeters of water, and weighs, empty, 29 grammes.

It is highly polished, and is kept so, especially on the bottom, so as to betray any particle of mercury that might soak through, during the determination, from the foil which is under it. However, this has not happened in a large number of assays, and is not likely to happen if proper care be taken in making the charge of ore. If it did happen, there would be no great harm done, as the mercury could easily be driven out of the dish by heat. This would involve, of course, the loss of that particular determination, which would have to be repeated with a smaller charge of ore.

The Retort.—As a retort, if I may call it so, I use a Battersea annealing-cup, size C. This is a crucible of unglazed white clay, in the form of a truncated cone, 2 centimeters outside diameter at the bottom and 3.5 centimeters diameter at the

mouth; the height is 4.5 centimeters. The mouth may be ground to an even surface in any convenient way, but generally this is not necessary.

Heat is applied to the bottom of the retort during the operation. To prevent the direct heating of the upper part of the crucible and of the silver-foil and cooling-cup, I make use of a circular tin shield, 13 centimeters in diameter, which has a hole in the center a little less than 3.5 centimeters in diameter. The annealing-cup will pass almost through this hole, and will remain firmly fixed therein, with about 1 centimeter of its upper part protruding.

The tin shield serves also to suspend the crucible and the rest of the apparatus from the ring of a retort-stand.

The annealing-cup used as a retort is of indefinite duration. The tin shield costs practically nothing.

Source of Heat.—For heating the apparatus I use a small glass lamp, which holds about 60 cubic centimeters of alcohol when comfortably filled. The brass wick-tube is 6 millimeters internal diameter, and when in action the wick is regulated to produce a flame from 4 to 5 centimeters high.

Flux and Charge.—For a flux I use iron-filings, the finer the better. Those I have in hand would go through a 60-mesh sieve. A long time ago I secured several hundred grammes of these filings, and prepared them for use as a flux by first lixiviating them with strong alcohol, to remove most of the grease, and then igniting the filings by heating them to redness for some time in a muffle. The filings are preserved for use in a glass bottle with a rubber stopper. The cost of this flux is practically nothing.

Probably litharge would do as well for a flux as the iron-filings; but I have found the latter quite satisfactory, and have never used any other. The litharge might melt and spoil the cup, which the iron will not do.

The charge for a mercury determination is from one-half to one gramme of ore. This is intimately mixed with 5 grammes of the prepared iron-filings. The mixture is accomplished in the annealing-cup itself by means of a spatula. About one gramme of iron-filings is placed on the charge as a cover.

The Operation.—After the crucible or retort has been charged as above described, it is hung by its tin shield from the ring of a retort-stand.

A piece of silver-foil is then cut large enough to cover the mouth of the crucible and leave a good margin of, say, one-half centimeter all round. The foil is carefully smoothed, and then ignited in the flame of the alcohol-lamp. Care must be exercised, for the foil is so thin that it will fuse at once if allowed to overheat.

After cooling sufficiently, under cover, the foil is weighed on an analytical balance, and is placed upon the mouth of the crucible. It is then gently pressed down with the finger until it moulds itself to the shape of the mouth of the crucible.

The cooling-cup is then placed upon the crucible on top of the silver-foil, and is filled with water.

The alcohol-lamp is placed under the crucible, and is arranged to give a flame about 4 centimeters high, which shall just barely spread out at its point over the central part of the bottom of the crucible.

The heating should continue in this way for from 10 to 15 minutes. Ten minutes is too short for most ores, and anything over 15 minutes is apt to lead to loss of mercury. Over 20 minutes is, in most cases, fatal.

The water in the cooling-cup may be renewed once or twice during the heating; possibly it would be well to use ice-water for this purpose. Water at ordinary temperatures, however, has always given me good results.

When the heating is finished, the crucible and contents are allowed to cool at least five minutes. When the silver-foil is removed, a distinct mercurial stain will be seen upon its lower surface, if there was the slightest trace of mercury in the ore.

The amount and depth of this stain is a rough indication of the amount of mercury in the ore. The foil is conveyed (under cover, to avoid dust) to the balance.

The increase in the weight of the foil shows the amount of mercury absorbed; and a simple calculation gives the amount of mercury contained in the original charge of ore.

In order to check the first determination, and to make sure that all the mercury has been collected, I repeat the heating on the same charge for about ten minutes more, and then weigh again. If the weight is constant, or there is a slight decrease, the amount of mercury obtained by the first weighing may be considered correct. If more mercury has been

absorbed on the second weighing I repeat the determination with a new charge and heat for a longer time, say from five to ten minutes longer, than at the first heating.

Accuracy.—I use a very good Kohlbusch analytical balance which weighs down to 0.05 milligramme; but I generally only weigh to 0.1 milligramme. With the above apparatus a content of 0.01 per cent. of mercury is clearly appreciable. The stain produced by that amount of mercury is plainly visible on the foil.

Owing to the extreme flexibility of the foil, a number or a date can be easily marked with a blunt point of any kind upon the corner of any one of the pieces of foil, and this can be preserved for future identification.

Some of these foils have been preserved in my laboratory for several years, and still show the mercurial stain without apparent alteration, and with no perceptible loss of weight.

On carefully igniting the foil over an alcohol flame the mercurial stain disappears, and the foil can be used for other determinations, probably for an indefinite number of times, if desired.

This test for mercury, viewed as simply qualitative, is more positive and more easily applied than most of the qualitative tests given in the books with which I am familiar.

Any prospector provided with a few ounces of fine iron filings previously ignited on a piece of iron over a stove, a small crucible or annealing-cup, an ounce of silver-foil, a copper cooling-dish and a spirit-lamp, can test his ores for mercury in a very easy way. When the spirit-lamp is not at hand, the necessary heat can be had from a lump of ignited charcoal.

With the addition of a fairly good pocket-balance and weights for weighing the charges, quantitative determinations can be made, the foils being reserved for future accurate weighing. In the laboratory, several copper cooling-cups can be kept, and a half dozen or more mercury-determinations can be made at one time, using as many pieces of foil. Not many assayers are rich enough to keep several gold covers on hand; but with the silver-foil there is practically no extra expense incurred by increase of facilities.

Failures.—These may arise from too high a heat or too long heating. Generally, however, they arise from the foil being

badly adjusted to the mouth of the annealing-cup. If this adjustment be carefully made, there should be no perceptible escape of mercurial vapor.

Some New Points.—I claim as original the use of silver for receiving the mercury. I am aware that this has been suggested before, but do not know that it has ever been carried out in practice.

The use of silver in the shape of thin foil, and the use of a separate vessel to cool the receiving surface by contact. By the combination of the thin foil and the cooling-cup the cost of the assay is materially reduced, and the foil can be kept for reference, if desired. All of this I believe to be quite new.

The surface of the pure silver-foil, if carefully ignited before exposure to the mercurial vapor, is extremely sensitive, and readily absorbs the slightest trace of mercury. The stain made by one one-hundredth of one per cent. upon the foil is as visible as the deposit of moisture left by breathing upon a window-pane.

Those analysts and assayers who may have been using gold covers will no doubt discard them after a trial of the present method, and I would respectfully suggest that all such covers should be sent to me, and I will undertake to see that their value is properly expended in the cause of science!

The Auriferous Deposits of Siberia.

BY RENÉ DE BATZ, PARIS, FRANCE.*

(Atlantic City Meeting, February, 1898.)

INTRODUCTORY.

FROM 1754 to the end of 1895 the production of gold in Russia had been approximately as follows:

	Kilogrammes.
Russia proper (Finland and the Caucasus),	390
The Ural Region,	505,386
Western Siberia,	116,937
Eastern Siberia,	1,218,372
Total,	1,841,085

of the value of more than $1,200,000,000.

* Translated by the Secretary.

For the five years ending with the close of 1895 the product of the Russian Empire was, on the average, 40,506 kilogrammes per annum, or 16.1 per cent. of the estimated total product (250,759 kilogrammes) of the world. In 1896 the Russian Empire produced 47,550 kilogrammes, or 14.54 per cent. of the total (327,081 kilogrammes).*

Although these figures indicate some falling-off in the proportion of the Russian production, they exhibit an advance in its actual amount. The percentage for any given period is, of course, affected by discoveries and developments in other countries quite independent of the Russian industry. Thus, in the decade preceding the discovery of gold in California and Australia, it amounted to 40 per cent. of the product of the world. Reduced by these events and their consequences to a small figure, it gradually recovered, and had risen above 20 per cent. when the development of the Transvaal, about 1890, again diminished its relative importance. It must be conceded, however, that the Russian Empire has contributed hitherto an enormous aggregate supply of gold, and that, in view of the fact that this supply has been produced without the aid of great immigrations of adventurers, large investments of capital, or modern methods and machinery, there must be behind it extensive and valuable natural resources, destined to prove hereafter still more productive than they have been heretofore.

Of the reasons for the slow and ineffective development of the gold-fields of Siberia, the following may be mentioned as the most influential:

1. The climate, with its excessively cold winter and short, almost torrid, summer, has not favored prolonged explorations. Moreover, its conditions have been exaggerated in popular belief. The investigations of French, German, English and American mining engineers during the past few years have, however, dissipated this impression to some extent, showing that, although the climate is severe, there is, in almost the whole of the inhabited and explored region, a summer of four or five months, with a mean temperature of 59° to 68° Fahr., and

* *N. Y. Eng. and Min. Journal*, January 2, 1897. For tables of statistics and detailed descriptions, the reader is referred to my book, *Les Gisements Aurifères de Sibérie*, Paris, 1896.

that the cold of winter is by no means unendurable. Nevertheless, there can be no doubt that such climatic conditions seriously handicap any industry; and though they may not be prohibitory, are expressed, in the case of mining, by a certain addition to the cost per unit of product.

2. Communications have been, and still are, difficult. Interrupted during three or four months by periodical snow-falls or thaws, they are, for the rest of the year, precarious, slow and uncomfortable to travelers. It takes two months to go from St. Petersburg to the Upper Amoor or the Zeia. But these conditions are undergoing gradual amelioration; and especially the rapid advance of the Russian Trans-Siberian railway will soon render a large part of eastern Siberia accessible for passengers and freight.

3. Labor, though at present relatively abundant, has not always been so; and the system of employment does not permit the rapid increase of a working-force at any one point.

4. Capital has always been lacking for explorations or installations looking to future recoupment. Operators have aimed at immediate revenue without the risk of considerable investments. Capitalists have been naturally disinclined to expend in such remote regions, through agents whom they could not efficiently watch and control, the money alleged to be necessary for investigations or preparations not yielding instant profit. Moreover, it may be said that, until recently, at least, Russia has been predominantly an agricultural country, without large manufacturing industries, and Russian capital has not been available for enterprises in new directions.

5. It should be added that the Russian, and especially the Siberian, is a man of routine, doing what his father did, and averse to novelties. The progress of the last forty years in the mining and metallurgy of gold has passed almost unnoticed by Siberian operators. A few feeble attempts to introduce Californian or other foreign methods and apparatus have failed, by reason partly of the hostility of the workmen, partly of the lack of competent instructors, and partly of the necessity for that adaptation to local conditions which no one on the spot was competent to make.

It is, however, safe to predict that the situation above outlined will be speedily changed. The gold-bearing districts,

penetrated by railroads and placed in communication by sea with Europe (Odessa to Vladivostok), Japan (Kobé to Vladivostok), and, before long, America (San Francisco to Nikolaievsk and Vladivostok), will become accessible to the enterprise, the science and the capital of the civilized world.

Plate I. is a mining map of Siberia, showing the districts of the three chief divisions of the Ural on the west, Tomsk in the center and Irkutsk on the east, and also of Turkestan on the southwest.

The subdivisions or mining arrondissements are numbered on the map as follows:*

The Ural.

I. Vyatka.
II. Perm.
III. West Ekaterinburg.
IV. Ufa.
V. Verkhoturie.
VI. East Ekaterinburg.
VII. Orenburg.
VIII. Orenburg South.

Tomsk.

1. Tobolsk-Akmolinsk.
2. Semipalatinsk-Semiretchensk.
3. Tomsk.
4. Northern Yenisei.
5. Southern Yenisei.
6. Atchinsk-Minousinsk.

Irkutsk.

I′. Primorskoi.
II′. Amoor.
III′. East Transbaikalia.
IV′. West Transbaikalia.
V′. Lena.
VI′. Birouzinsk.

O. Arrondissement of Turkestan.

* For a detailed account of the geographical division into districts, and of the mining laws, regulations and official administration, I must refer the reader to my book, already cited.

The very incomplete and general statements of this paper will deal chiefly with eastern Siberia.

General Geology of the Siberian Placers.

The difficulty of a systematic geological study of the auriferous alluvions of Siberia is due to their immense extent and to the almost impenetrable virgin forest, with its abundant vegetation, fallen timber and swampy undergrowth, which prevents, in most cases, the tracing of strata or vein-outcrops or the identification of the underlying rocks. A general statement on this subject must be, therefore, necessarily imperfect and provisional.

Unlike the great deposits of the Rocky Mountains in California and British Columbia, which are proved by the lavas which cover them to be pre-Tertiary, the Siberian gold-bearing alluvions are Quaternary, and of recent formation. (The alleged recognition of more ancient placers lacks, as yet, in most cases, thorough investigation and confirmation.*) Occupying valleys usually of gentle slope, they are characterized by the fact that the pay-stratum (*plast*), being on the bed-rock (*potchva*), is covered, in the immense majority of cases, by a barren layer called *torf*. (This term, meaning turf or peat, owes its present wide application to the circumstance that the first placer, discovered in the Ekaterinburg district, was covered with a bed of real peat.)

The auriferous regions generally show low, rounded hills, bearing witness to the destructive action of atmospheric agencies upon the rocks.

The placers are above sea-level: in the Ural, 150 to 300 meters; in the Alataou, 600 meters; in the valleys of the Olekma, and in the Yenisei, from 600 to 750 meters.

Besides native gold (often in nuggets) they contain pyrite (frequently mispickel), and all the products of its decomposition—magnetite, limonite, hematite, etc. Copper occurs, sometimes native, sometimes as chalcopyrite. Lead has been found

* M. Obrontcheff, however (*Mémoires de la Section de la Sibérie Orientale de la Société de Géographie Impériale Russe*, vol. xxiii., 1892), asserts in the region of the Olekma the existence of deep, pre-glacial placers, exposed particularly in the basins of the Ougakane and Kadalikane rivers. Ancient placers have been described also in the basin of the Zeia by the geologist Makéroff.

as sulphate, carbonate, phosphate and sulphide. Native bismuth has also been found. Cassiterite, garnet, rutile, tourmaline, zircon, etc., should be added to the above list.

Of fossil remains, the most numerous have been those of the mammoth (*Elephas primigenius*). Already, in 1840, it was estimated that more than 20,000 of these animals had been exhumed. Some of them, still retaining the soft fleshy portions of their bodies, have been found in the frozen clays of the extreme North. The preservation of these animals, which must have lived in the midst of an abundant vegetation, is a striking proof of the rapidity with which the glacial period spread over the great plains of Siberia.

Besides the bones of other mammals, now extinct, human remains have been found in the Siberian alluvions: a skull, encountered in 1860, at the depth of 3 meters, in the Tchtogolev mine, and fire-places, together with a stone slab, covered with inscriptions, in the Proroko-Illinsky mine, in the Kigass basin.

A peculiar characteristic of the Siberian placers, especially those of the *arrondissement* of the Lena, the district of the Zeia, etc., is that the soil is perpetually frozen, or can be thawed only with great difficulty in the summer.* Near the head-waters of the rivers, however, there are sometimes found places (called *talik*) which are not congealed, and the occurrence of which is believed to be due to subterranean tepid or hot springs.

Special Geology.—In the Altai region, the predominant rocks outcropping or exposed as bed-rocks are sandstones and argillaceous schists, with which occur metamorphic rocks, approaching granites and diorites.

The dominant rocks in the districts of Yenisei, both North and South, are metamorphic schists, principally argillaceous, and passing, in some places, into mica-schists. In the northern, more frequently than in the southern district, granites, gneisses, diorites and porphyries are encountered, with sandstones and conglomerates here and there. The auriferous sands generally occur in the schists, along their lines of contact with amphibolic granites or diorites.

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* NOTE BY THE SECRETARY.—This is also a characteristic of part of the auriferous region in Alaska and the Yukon district of the Northwest Territory of the Dominion of Canada, now popularly known as "the Klondike country."

R. W. R.

In the province of Yakutsk there is a strong formation of granite-syenite, passing at places into gneiss, which in turn passes into micaceous, chloritic, talcose and argillaceous schists.

In the district of Nertchinsk, granites, gneisses, syenites and diorites have been encountered.

Finally, in the basin of the Amoor the most frequent rocks are micaceous and amphibolic gneisses, mixed with schists, into which they pass by insensible gradation.

Economic Conditions and Value.

The auriferous sands of western Siberia and Yenisei are very much poorer than those of eastern Siberia. The first-named carry 20 to 30 *dolis* per 100 *poods* (say 60 to 90 cents per cubic yard). They are limited in size; and the pay-stratum is thin, irregular and discontinuous. The placers of the "governments" of the Steppes and of Tobolsk are the poorest (yielding respectively, in 1895, an average of 12.8 and 20.58 *dolis* per 100 *poods* (say, 38 and 66 cents per cubic yard).

The placers of the Yenisei have been celebrated for their richness, and for the thickness, continuity and uniformity of their pay-stratum; but the richest portions have been entirely exploited, and they now support but a petty industry.

The thickness of the *torf* varies from 1 *archine* (0.7 m.) to 5 *sagènes* (10.65 m., or 35 feet); the pay-gravel (*plast*) is from 0.5 to 3 *archines* (0.35 to 2.15 m., or 7 feet) thick. An exceptional thickness of *plast* is reported from certain placers of the Ogné river, where it exceeds 20 *archines* (14.20 m., or 46.6 feet). In the mines of the Astacheff Co. it is said to be 15 *archines* (10.65 m., or 35 feet) deep. The width is usually 15 *sagènes* (32 m., or 10.5 feet); sometimes, as on the Enatchimo river, it reaches 100 and 200 *sagènes* (700 and 1400 feet). The grade of the rivers varies from 0.25 to 6 *verschoks* to the *sagène* (5 to 125 millimeters to the meter; 25 to 625 feet to the mile). In 1895 the average richness of the sands was, for the northern Yenisei district, 28.33 *dolis* per 100 *poods* (85 cents per cubic yard); for the southern district, 27.30 *dolis* per 100 *poods* (82 cents per cubic yard).

On the other hand, the placers of the district of Nertchinsk, the country of the Amoor river and the department of the Lena are remarkably deep, extensive and continuous.

In the department of Nertchinsk or East Transbaikalia, three systems or groups of placers have been distinguished: that of the Chilka, which is the most westerly and the richest; the central group, extending to the sources of the Onone, and the eastern group, which is the poorest. The tenor of gold averages from 60 *dolis* to 1 *zolotnik* per 100 *poods* ($1.80 to $2.88 per cubic yard). The Kara placer presents the high average last named, but is now nearly exhausted. The pay-stratum is 2 to 4 *archines* (4.6 to 9.3 feet) thick; and the richest parts carry as much as 2 to 3 *zolotniks* per 100 *poods* ($5.76 to $8.64 per cubic yard). It is in this district that placers are worked on a large scale, and here also that drift-mining is practiced under a heavy barren over-burden.

In the Amoor basin, the average thickness of the barren cover is 1 *sagène* (7 feet), and that of the pay-sands about one-half as great. All the claims are easily worked by open cut, except on the river Nimane, where some underground workings are found; the thickness of the barren cover being 20 feet, and that of the pay-stratum 9 feet.

The alluvions of the Lena basin carry two and sometimes three pay-horizons. In the Olekma district, the average value is $4.24 to $5 per cubic yard; in that of Vitim, it runs from $8.50 to $13.45; and in the Predtechensky mine of the Company of the Lena, it has even reached $19.61. The gold is coarse, frequently in nuggets, and often crystallized. On the other hand, the thickness of barren cover is often 10 *sagènes* (70 feet), and reaches in places even double that thickness. Moreover, both the over-burden and the pay-stratum are perpetually frozen.

Deposits in Rocks.

Veins have been worked very little in Siberia. Nevertheless, they are not unknown. In a large number of placers they have been found traversing the bed-rock which had been exposed by mining. They carry gold generally in pyrites. In Altai, Yenisei and Transbaikalia, quartz-mines have been worked in a small way.

Upon the river Ili occurs the peculiar auriferous deposit of Evdokie-Vassilievski, in a promontory about 75 meters above the stream. The mass of this promontory is porphyroidal biotite-granite; the length of the deposit is 700 feet and its width

490 feet. The gold is disseminated in little grains or needles through the granite, which passes sometimes into aplite (mica-less granite). The granite contains also pyrite, manganese oxide and malachite in small quantities. It is especially rich in gold in the vicinity of two quartz-porphyry dikes, which are 35 to 42 feet thick and 175 feet apart. The yield of gold has varied between $4.28 and $12.56 per ton.

Origin of the Placers of Eastern Siberia.

The Siberian placers are found both on the north and on the east of the great elevation which traverses Asia from northeast to southwest. This vast plain, cut by important rivers (the Selenga-Angara-Yenisei, 4750 kil., the Ob, 4230 kil., the Lena, 4040 kil., and the Amoor, 4360 kil. long), shows the traces of a considerable glacial activity.

The genesis of the Siberian placers, therefore, may be assumed to have been analogous to that of the similar deposits studied in the United States, Europe, Australia, etc. The Silurian or Devonian sediments, elevated at the epoch of the formation of the grand Asiatic chain, would have been fissured and broken. The fissures, filled with quartz and pyrite, derived from the schists by circulating waters, would become the veins which are already recognized or will be hereafter discovered in the mass of the mountains. The powerful meteoric and mechanical agencies of the glacial period, grinding down hills, filling valleys and leveling the vast boreal plain, would give rise to the placers which are now mined.

If this hypothesis be correct, then the original deposits, the "mother-lodes," are to be sought, in many cases, to the south of the localities now exploited.

It is already known that the masses of the Altai mountains contain deposits of lead, silver and copper, and samples of all kinds of ores are brought in great number by hunters and foresters. It is in these masses and their westward prolongation, and in the mountain-complex which extends northeast from Manchuria, that the permanent wealth of Siberia must be sought. The Khingane chain (called by the Chinese "the Mount of Gold") is said to be rich in minerals; and it is from this range and its buttresses that the gold of the Oussouri and lower Amoor placers must have come. Similar things are true of

the ranges which furrow the high plateau of the Vitim, and of those which border the Sea of Okhotsk. These are the sources of Siberian gold; it is in these that, in a not distant future, an exploitation will be established more permanent than that of the alluvions, and more effective in promoting the colonization and the enrichment of the country.

Labor and Wages.

Unlike the United States and Australia, this region (eastern Siberia and a part of western Siberia) possesses no permanent settlements at the placers. The workmen go to the gold-fields for the campaign (or "operation" as it is called in Siberia) of about four months, and go away when it is over. They come May 18th and go September 10th (old style), leaving only the staff of the management, certain mechanics, and a few laborers who strip *torf* during the winter.

The miners proper, employed in handling and washing the auriferous gravel, are hired in the more populous parts of Transbaikalia by agents, who execute written contracts with them, advance them on account 15 to 50 paper roubles each, and send them in barges to the gold-fields as soon as navigation opens. This involves, of course, a considerable loss of working-time. Wages are a first lien on the gross product of gold. The workmen are lodged and fed by the company, in rude barracks. About one-quarter of them are accompanied by their wives, some of whom are employed as cooks, laundresses, nurses, etc., while the rest pay a trifling sum for their board. The company maintains a hospital, and furnishes medical attendance free. It also keeps a store, in which the prices of goods are fixed by the government. The sale of alcohol and its possession by the workmen is strictly forbidden; but a daily allowance of 15 to 20 centiliters of *vodka* (about equal parts of alcohol and water) is served daily to each laborer (or withheld as a penalty for violations of rules, etc.).

Salaried employees and mechanics are paid differently in different localities. The following figures obtain in the placers on the tributaries of the lower Amoor, in the Maritime Province.

Salaries range, according to duties, from 700 to 1200 roubles per annum.

Special workmen are paid by the month as follows:

Employment.	Winter. (Oct. 1 to May 1.) Roubles.	Summer. (May 1 to Oct. 1.) Roubles.
Stable-men,	24	45
Bakers,	30	50
Sieve-men,	15	30
Watchmen,	18	30
Cooks,	40	40
Water- and Wood-carriers,	—	30
Male servants,	18	30
Female servants,	12	12
Laundresses,	10	10
Blacksmiths, 1st class,	40	60
Blacksmiths, 2d class,	36	45
Blacksmiths' assistants,	24	40
Joiners,	30	50
Carpenters,	30	45
Harness-makers,	24	40
Masons for stoves,	24	36
Dump-foremen,	24	30
Road-laborers,	—	30
Dischargers at the machine,	—	57
Tailings-foremen,	—	30
Hopper-foremen,	—	40
Washers on the table,	—	50
Washers on the table,	—	45

The miners work by the piece in *artels* or gangs formed by themselves, under chiefs of their own choosing. Each gang comprises 7 men, and is furnished by the company with three barrows and the necessary tools. According to the contract, the *artel* must move a daily minimum of 5 cubic *sagènes* (63.6 cubic yards) in winter and 7 (89 cubic yards) in summer, of *torf*. When working upon sand or gravel, it must carry to the washing-machine from 4 to 6 cubic *sagènes* (54 to 76 cubic yards) daily, according to the character of the material. They are paid:

For *torf*, 1.25 rouble per cubic *sagène* for the minimum amount and 3 roubles for each cubic *sagène* in excess.

For sand, 1.35 to 2.65 roubles for the minimum, and 4 roubles for the excess, per cubic *sagène*.

Work begins in summer at 4.30 or 5 A.M. From 7.30 to 8 a rest is taken and tea is served by the company. From 11 to 1, dinner and a rest in the barracks; from 3.30 to 4, tea again. The work ends between 7 and 8 P.M. There is no night-work.

Methods of Working.

Prospecting (*Poiski*).—This is carried on by organized expeditions of considerable size. Single prospectors, or small parties of two or three, cannot explore the inhospitable, swampy, uninhabited forest-wilderness with advantage. Prospecting-expeditions are usually sent out by syndicates, under skilled leaders, and are provided (in eastern Siberia) with native Mongolian guides, teams of horses, reindeer or dogs, tools, camp-equipage and provisions. They start when winter has set in, and the freezing of the ground has made the swamps passable, say, in October or November.

Geological indications there are none, for the whole country is covered with snow at this season; but the topography, the direction, grade and general aspect of the valleys, etc., furnish some guiding signs to an experienced leader. Still, the selection of a favorable spot is largely a matter of accident. There is a classic story of one of these chiefs, who went on a prolonged "spree" while his party was at work; and who, although the *torf* was found very thick, and the gravel beneath it barren, persisted with drunken obstinacy in his order to "go on digging," until at last, to the amazement of his men, one of the richest pay-strata in Siberia was struck, at the depth of over 200 feet—a depth to which no sane and sober man would have pushed an unpromising exploration of the kind.

A place deemed favorable having been reached, the party goes into camp, and begins its examination, under the ice, in the bed of the stream. If gold be found, prospecting-pits are sunk in the frozen soil of the valley, and the sand obtained is crudely washed, either with the *batea* or on an inclined table common in Siberia, and called a *vacheguert* (a corruption of the German *Wasch-heerd*).* In remote districts, and where pumps are not available, this work is preferably done in the winter,

* The *vacheguert* is an inclined plane about 2 meters long and 1 meter wide, upon the upper end of which water is distributed in a sheet. The material, deposited upon the upper part of the plane, is stirred with a wooden hoe or scraper, so as to push it against the current, until the lighter portions have been washed away, and only such substances as gold, pyrite, iron oxides and heavy silicates are left. The operation is then more carefully continued with a trapezoidal wooden scraper, and finally with a soft brush, until almost nothing but pure gold remains. This is collected on a small sheet-iron shovel and dried at a gentle heat.

when the ground is frozen. (Frequently, however, the neighborhood of hot springs nullifies this effect of the season.) If the results are sufficiently encouraging, posts are set and a *zayavka* (corresponding to an American notice of location) is prepared, as a basis of the application for mining rights; after which the expedition returns or proceeds to further explorations, as the case may be. It is estimated that the cost of such an expedition in eastern Siberia is 800 to 1000 roubles per man employed. The leader of the party receives generally a part of the subsequent profits of the undertaking. (A common figure is one-sixtieth of the gross product; sometimes it is 200, 300 or even 400 roubles for each *pood* of gold won—the *pood* being worth 18,000 roubles.)

Preliminary Testing (*Razvietki*).—This operation, which has for its object the determination of the paying portions of a placer (as Americans would say, the pay-channel), follows rules which obtain throughout Siberia, from the Ural to the Pacific. The surface of the claim is covered with prospecting-pits 10 (or even only 5) *sagènes* (70 or 35 feet) apart, on lines 20 to 30 *sagènes* apart, crossing at right angles the axis of the valley. These pits are 7 by 7 feet in size, and are designed to reach bed-rock (*potchva*). After passing through the soil and clay overlying the sand and gravel, the material extracted is deposited in samples, each comprising one *archine* (2.33 feet) of depth, which are afterwards washed. Careful records being kept of the position and results of each pit, the whole investigation furnishes a guide to further operations.

Stripping.—The first step, after felling the trees and extracting the stumps, is to lay bare the *plast* or pay-stratum. This is done in benches about 1 *archine* (2.33 feet) high, with pick, shovel and crow-bar, and one-horse two-wheeled carts (*tarataikas*) of the capacity of 1000 to 1500 pounds. The material is dumped either on a part of the placer already worked out, or on ground supposed (sometimes erroneously) not to overlie possible future workings. The law requires that the dumps (*otvals*) of the barren over-burden shall be separate from those of the tailings (*efeli*). The total cost for labor, superintendence, horses, etc., of moving to the dump one cubic yard of material is reported to be, in eastern Siberia, about 18 cents.

Mining of the Pay-Stratum.—This is done in the same way as the stripping. On the Amagoune, a gang of 7 men carries to the washer in a day (10 hours) 190 cubic yards of pay-stuff, at a total cost for wages, horses, superintendence and repairs, of 5.29 roubles, or, say, 23 cents per cubic yard. Of this amount, 13 cents is the cost of the crude and expensive transportation to the washing-sluice (1000 to 1500 feet), involving the labor of 3 men to every gang of 7 miners. This is the weakest point in the economy of mining in Siberia, particularly in eastern Siberia.

Washing.—The essential apparatus used for this purpose in Siberia, from the Ural to the Pacific, is the *schlouss* (analogous to the American sluice), an inclined table of moderate length and relatively steep grade, provided with riffles to receive the gold, and supplied with water by wooden conduits. At the lower end, pyramidal boxes receive the washed sands, while the water and lighter materials escape into the valley.

The simple sluice is often provided with mechanical means for breaking up the material, such as a drum, driven by an undershot water-wheel. A machine of the latter kind will wash in two hours about 2000 cubic feet of gravel.

The material caught in the riffles is removed twice a day to the *amerikanka* (a small secondary sluice with iron riffles) and washed again; this second concentrate being finally treated on the *vacheguert*, already described. Such a machine (with drum, etc.) costs, installed, about 7000 roubles; and the cost of each removal of it required by the progress of the work (increasing the distance of transportation for the material) is about 3000 roubles. From gravel containing fairly coarse gold, and free from float-gold, the machine saves about 85 per cent. of the value; 90 per cent. of this saving being made on the upper 14 feet of the sluice. The total cost of washing (labor, superintendence, repairs and interest) is about 4.8 cents per cubic yard.

Drawings of several types of washing-machines used by leading Siberian companies are given in my book, already cited, to which those desirous of more detailed information are referred.

Special Methods of Working.—Drift-mining is sometimes practiced when the *torf* is too thick for economical stripping and

open-work. In these cases shafts, about 175 feet apart, are sunk to bed-rock along the valley, and connected by a main gangway, from which cross-cuts are run to the limits of the pay-channel. From these, again, breasts (usually about 14 feet wide, with intervening pillars of the same width, left for subsequent removal) are driven parallel to the main gangway. This work is done in the winter, the whole force being employed during the summer in washing. But the ground, though often frozen, requires costly timbering.

Dredges, such as are used in New Zealand, have been tried by several companies, but without success, by reason, partly of the opposition of the workmen to such innovations, partly to the lack of machine-shops near by, and the consequent cost and delay of repairs, and, perhaps chiefly, to the fact that an apparatus suited to one locality cannot be expected to suit, without modification, the special conditions of another.

The same general observations may be made concerning the use of mechanical excavators, which have had the additional disadvantage that they delivered to the washing-machine masses larger than it could treat, or the treatment of which required an increase in the supply of water, exceeding in cost, under the circumstances, the saving to be effected by mechanical excavation.

Appendix.

The following tables are given in the belief that they may be useful to members of the Institute in the study of reports of Siberian mining, which will not improbably be more frequently brought to their attention hereafter than heretofore:

Table I.—*Russian Measures and Their Metric and English Equivalents.*

Measures of Weight:

1 Berkovitz = 10 poods = 163.8 kilos.
1 Pood = 40 livres = 16.37963 kilos.
1 Livre = 32 lots = 96 zolotniks = 0.40949 kilo.
1 Lot = 3 zolotniks = 0.012797 kilo.
1 Zolotnik = 96 dolis = 4.2653 grammes.
1 Doli = 44.5 milligrammes.
100 Poods = 1637.963 kilos = 1.88 English short tons of 2000 pounds avoirdupois.
1 Zolotnik = 0.137 ounce Troy.
1 English short ton = 907 kilos = 55.31 poods.

1 Ounce Troy = 31.104 grammes = 7.29 zolotniks.

Measures of Length:

1 Russian mile = 7 versts = 7.467 kilom.
1 Verst = 500 sagènes = 1.06678 kilom. = 3500 feet.
1 Sagène = 3 archines = 7 feet = 2.13356 meters.
1 Archine = 16 verschoks = 0.711187 meter.
1 Verschok = 0.044449 meter.
1 Russian foot = 1 English foot = 12 duims = 0.30749 meter.
1 Duim = 1 English inch = 0.0254 meter.
1 Tchetvert (used in Siberia) = 0.25 archine = 0.17779 meter.

Measures of Area:

1 Deciatine = 2400 square sagènes = 1.09252 hectare.
1 Square sagène = 4.552078 square meters.

Measures of Volume:

1 Cubic sagène = 27 cubic archines = 343 cubic feet = 9.712 cubic meters.
1 Vedro = 12.29891 liters.

TABLE II.—*Russian, Metric, English and American Measures of the Gold-Value of Auriferous Sand or Gravel.*

Per 100 Poods. Dolis.	Per Metric Ton. Grammes	Per Eng. Short Ton. Oz. Troy.	The same in U. S. Gold. Dollars.	Approx per Cub M. or Cub. Yd. (about 100 Poods). Weight, Grammes.	Approx per Cub M. or Cub. Yd. (about 100 Poods). Value, Dollars
1.	0.027	0.00078	$ 0.016	0.044	$ 0.03
2.	0.055	0.00156	0.032	0.089	0.06
3.	0.082	0.00234	0.048	0.132	0.09
4.	0.109	0.00312	0.064	0.178	0.12
5.	0.136	0.00390	0.080	0.222	0.15
6.	0.163	0.00468	0.096	0.266	0.18
7.	0.190	0.00546	0.113	0.312	0.21
8.	0.217	0.00624	0.129	0.356	0.24
9.	0.244	0.00702	0.145	0.405	0.27
Zolotniks.					
1.	2.604	0.076	$ 1.57	4.265	$ 2.83
2.	5.208	0.152	3.14	8.530	5.67
3.	7.812	0.228	4.71	12.795	8.50
4.	10.416	0.304	6.28	17.060	11.32
5.	13.020	0.380	7.85	21.325	14.17
6.	15.624	0.456	9.42	25.590	17.00
7.	18.221	0.532	10.99	29.855	19.84
8.	20.832	0.608	12.56	34.120	22.66
9.	23.436	0.684	14.13	38.385	25.50
13 zolotniks and 15.5 dolis	34.271	1.000	$20.67	....	....

Graphic Records of the Screening of Crushed Materials.

BY PROF. COURTENAY DE KALB, THE SCHOOL OF MINING, KINGSTON, ONT.

(Buffalo Meeting, October, 1898.)

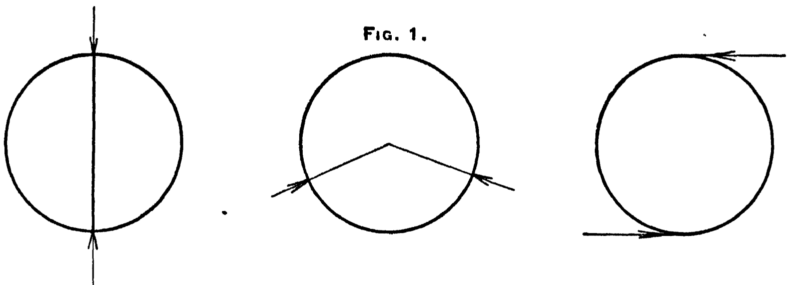
So far as the writer is aware, no detailed investigation into the behavior of ores or rocks when subjected to crushing under different conditions has yet been made. He cannot himself claim to have carried such investigations very far; but the results of some studies which he has been pursuing for a couple of years, as opportunity permitted, seem sufficiently interesting and suggestive to warrant a brief statement of them, preliminary to undertaking a more exhaustive course of experiments. Criticisms and suggestions are earnestly invited, with a view to making these studies as instructive as possible.

As a part of the examination of ores, preparatory to concentration, my students have been required to take lots of 100 pounds each, and crush them. These are then screened through a series of hand-sieves, 15 inches in diameter. The products remaining on each screen are weighed, and from the results is plotted a curve, called the sizing-curve. It was assumed at first that different ores would yield, under like conditions of crushing, curves showing unlike characteristics as to the quantities remaining on the various sieves. After a number of such curves had been obtained it became evident, as often happens, that this *a priori* reasoning was faulty. The curves presented practically identical characteristics, and only by varying the crushing-conditions were variations in the curves obtainable. Another peculiarity, which was as little anticipated, appeared by inspection of these results, namely, that successive crushing to smaller sizes did not greatly augment the quantities of the finer products, so long as the machines employed applied the crushing-force to the ore-particles in the same manner. In these experiments the fines were not eliminated between successive crushings.

I believe it will be readily admitted that rock can be crushed

by the application of mechanical force in only three ways: first, by forces acting in opposite directions along a diameter of the particle (*a*, Fig. 1); second, by forces acting along radii making an angle with each other (*b*, Fig. 1); and third, by forces acting in opposite directions along parallel lines, respectively tangent to the ore-particle at the two extremities of a diameter (*c*, Fig. 1). For convenience we may call the first *diametral crushing*, exemplified in the stamp-mill. The second we may call *radial crushing*, the principle most widely employed in crushing-operations, of which the reciprocating jaw-crushers, gyratory crushers, and rolls are our most familiar types. The third is generally recognized as *shearing*, and is exemplified in disk-grinders, cone-grinders, and similar types of mills.

FIG. 1.



a b c

Diagram Illustrating Different Methods of Crushing.

Two or all of these principles may be applied simultaneously in the same mechanism, as, for example, in the Chilian mill, where the revolution of the roller induces radial crushing upon first contact with an ore-particle, followed by diametral crushing due to the great weight of the roller, and this, finally, accompanied by shearing, as the roller is twisted from one tangential direction to the next succeeding one. Ball-mills also are compound crushers, although for the most part the work in these is done by radial crushing. After a sufficient quantity of data has been accumulated it would appear to be possible to test the action of any crusher by comparison of the curves from ore reduced in it with other curves in which the method of crushing was open to no doubt.

A selection has been made from a large number of curves for illustration in this paper. In each of these cases 101 pounds of material were taken, an allowance of 1 pound

being made for losses in handling. The resulting products aggregated in some instances a trifle more and in others a trifle less than 100 pounds, and the weights of ore on the various screens were accordingly taken as being percentages. The samples were as follows:

A. A coarsely crystalline syenite, containing mainly orthoclase feldspar, with soda and lime feldspars, finely divided muscovite, and small quantities of basic silicates. The relatively heavy bodies consisted of 27.94 per cent. of corundum in well-defined crystals and crystalline masses, and 4.64 per cent. of

FIG. 2A.

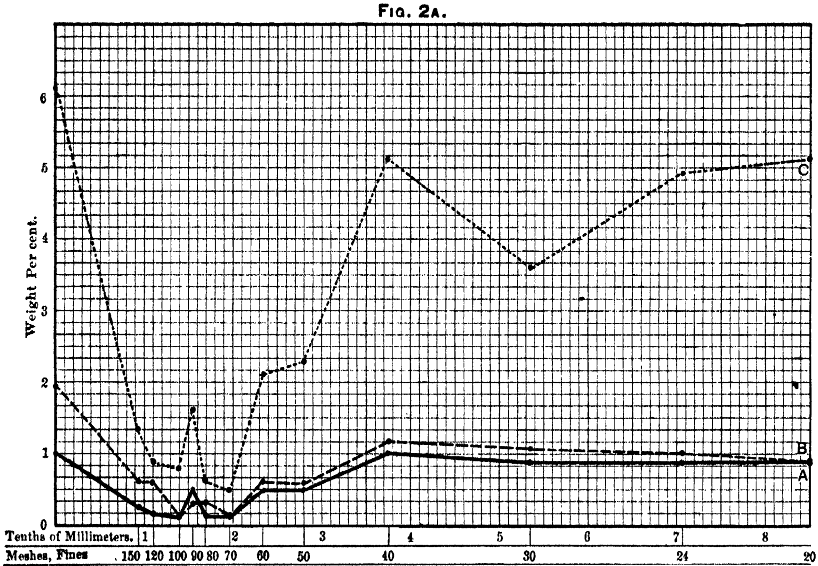


Diagram Illustrating Elevation of Curves, without "Sliming," by Successively Finer Reduction by Radial Crushing of the Same Rock.

magnetite. Crushed once in a Blake crusher, with maximum opening between jaws of 1 inch.

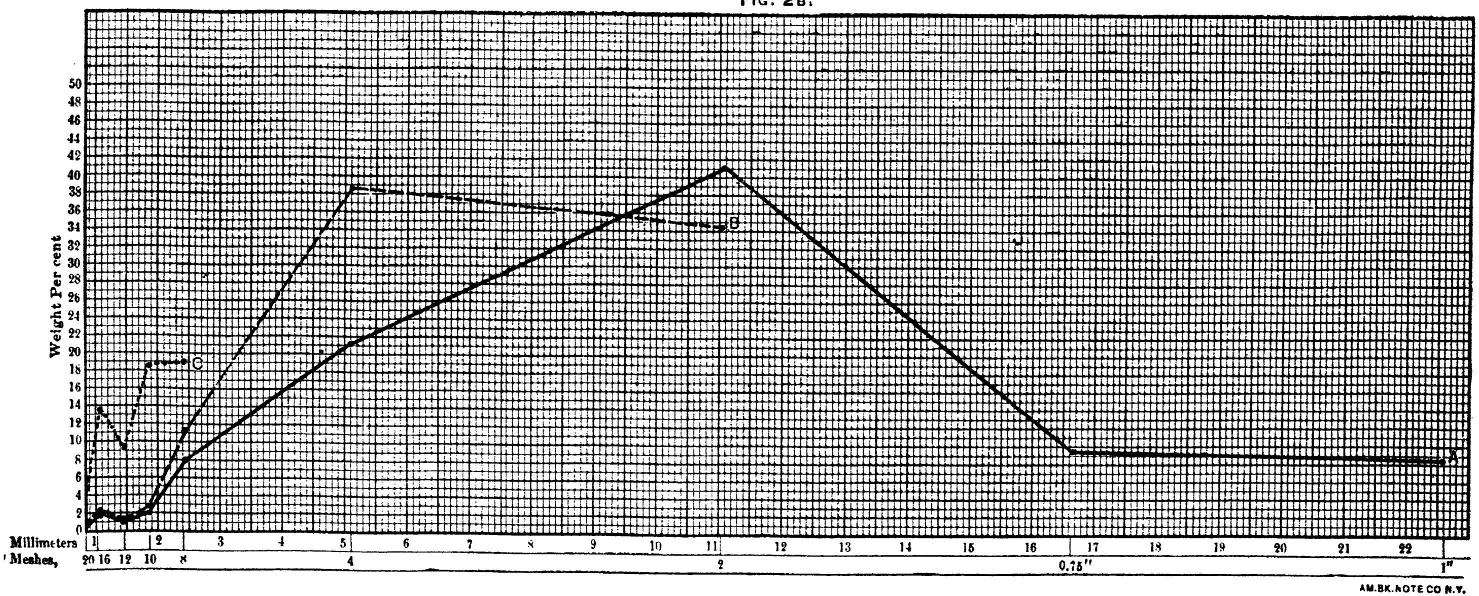
B. Same as A. Crushed first in a Blake crusher to 1 inch, and then in rolls set $\frac{3}{8}$-inch apart.

C. Same as A. Crushed first in a Blake crusher to 1 inch, then in rolls to $\frac{3}{8}$-inch, and finally in rolls to 0.2 inch.

D. True vein-material, consisting mainly of crystallized calcite, with small quantities of quartz, carrying 14 per cent. of chalcopyrite. Crushed once in a Blake crusher to 1 inch.

E. Same as D. Crushed first in a Blake crusher to 1 inch, and then in rolls to $\frac{3}{8}$-inch.

FIG. 2B.



Continuation of Fig. 2A, on Reduced Scale. Showing also Effect of Re-Crushing to a Maximum Size Corresponding to the Size of Mesh on which a Culminating-Point Occurs on the Curve from a Former Crushing.

FIG. 3.

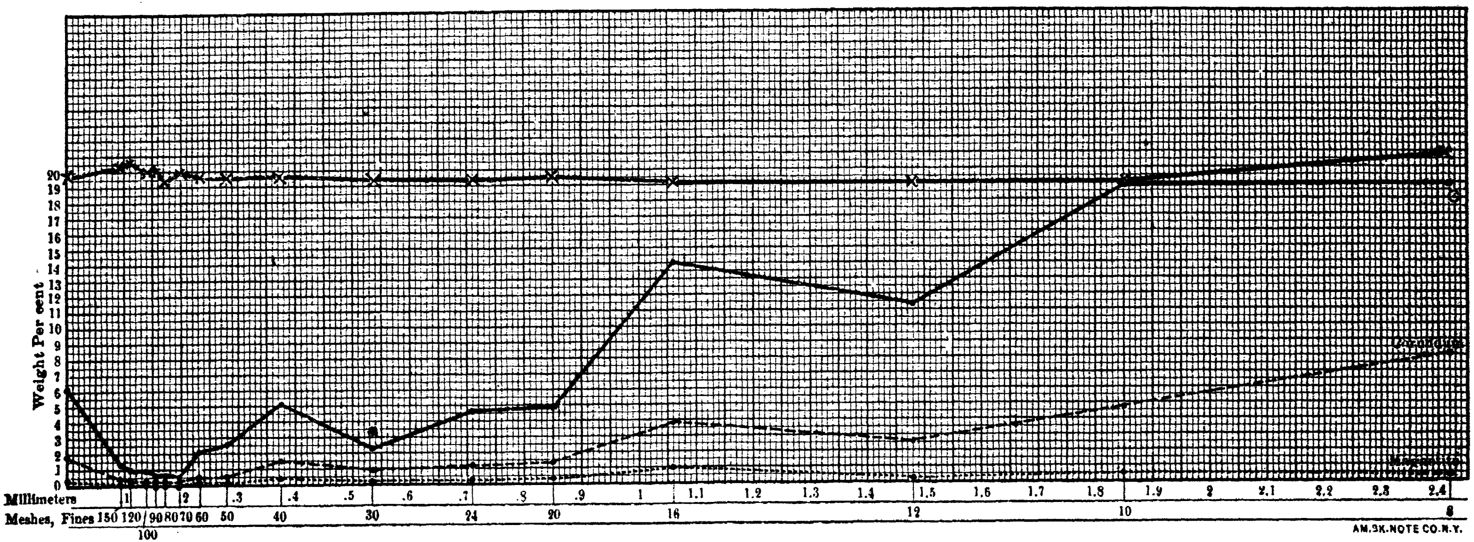


Diagram Illustrating the Distribution of the Relatively Heavy Minerals through the Various Sizes of a Crushed Rock. The Upper Curve Represents the Specific Gravities of the Whole Samples in each Size, beginning with a Specific Gravity of 3.08 on the Right-Hand Side.

F. Same as D. Crushed first in a Blake crusher to 1 inch, then in rolls twice to No. 8 mesh (2.42 millimeters), and finally in a cone-grinder (Fraser and Chalmer's sample grinder) to No. 20 mesh (0.85 millimeter).

G. Crystalline quartz-ore, containing a small (but undetermined) percentage of auriferous pyrites. Crushed in a stamp-mill, using a No. 30 mesh screen. The fact that a small quantity of the tailings remained on our No. 30 mesh-sieve shows that the diameter of openings in the stamp-mill screen was larger than in the sieve used for our screening.

FIG. 4A.

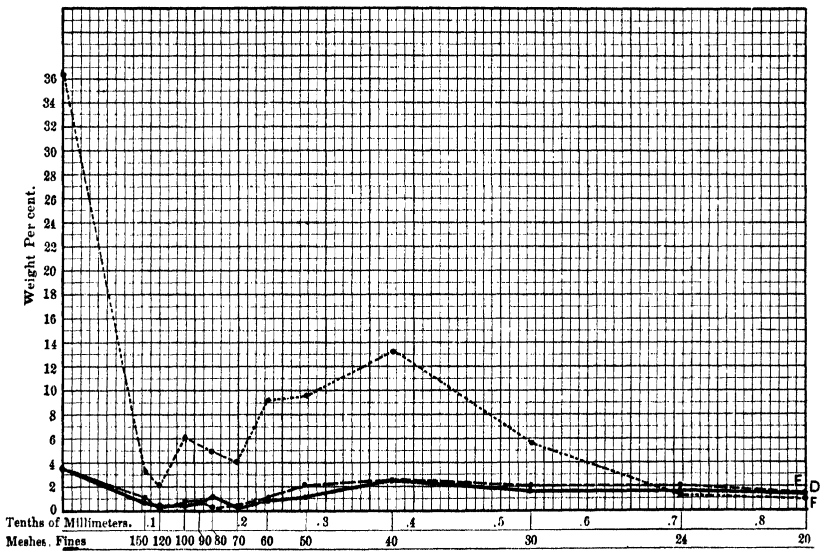


Diagram Illustrating Effects of Successively Finer Crushing of the Same Rocks. Curves D and E Represent Reduction by Radial Crushing (Blake Crusher and Rolls), and Curve F by Shearing (Cone-Grinder).

H. The characteristic cherty-lime ores of the Joplin, Mo., lead- and zinc-district. Crushed in a Dodge crusher to 1½ inch.

I. True vein-material, consisting almost wholly of crystalline aggregates of pink dolomite, with 15 per cent. of galena, the latter being in aggregates of minute cubes, and in part disseminated through the dolomite masses. Crushed to 1 inch in a Blake crusher.

J. Vitreous quartz, with a considerable quantity of disseminated pyrite. Crushed in a Blake crusher to 1 inch.

K. Vitreous quartz, with 47 per cent. of galena, in compact masses for the most part, but also disseminated to some extent. The quartz showed a remarkable tendency to break into greatly elongated particles. Crushed first to 1 inch in a Blake crusher, and then to $\frac{3}{8}$-inch in rolls.

L. Same as K. Crushed first to 1 inch in a Blake crusher, then to $\frac{3}{8}$-inch in rolls, and finally to 0.1 inch in rolls.

M. A very hard, tough, bluish-gray quartz-schist, containing fine, disseminated pyrite. Crushed to 1 inch in a Blake crusher.

N. Coarsely crystalline dioritic rock, the feldspar mainly oligoclase and albite. Much muscovite, and small quantities of basic silicates. The heavy bodies consist mainly of large crystalline masses of corundum and magnetite.

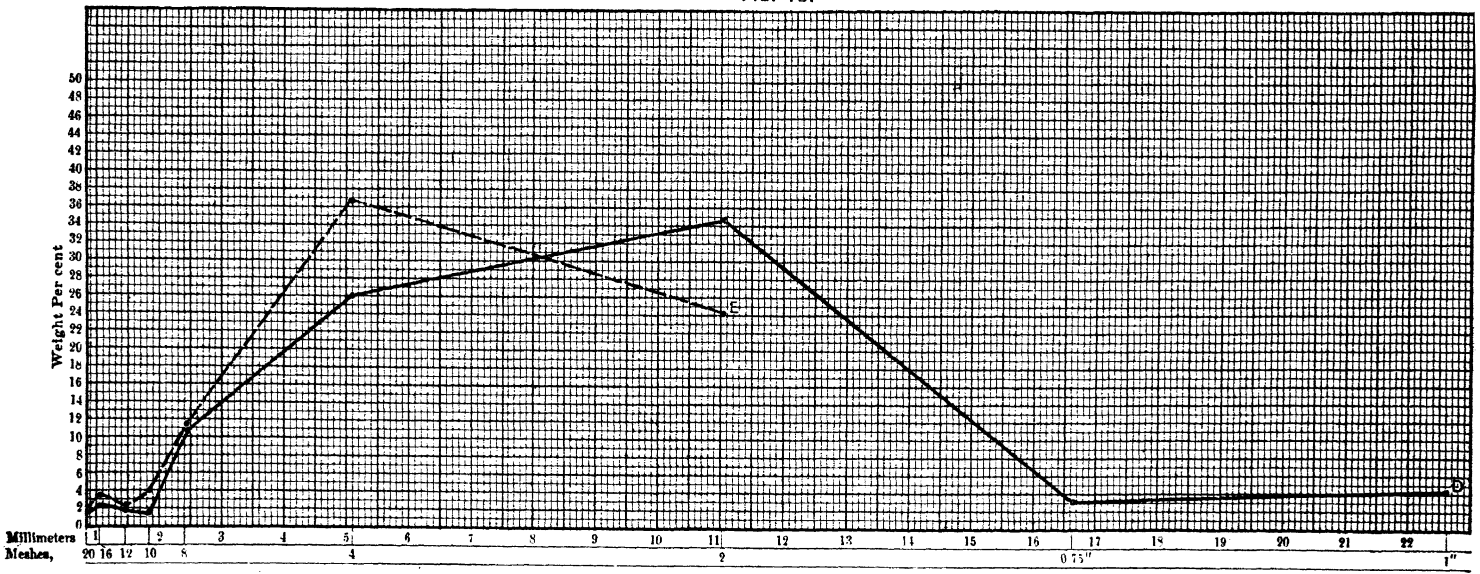
An examination of the diagrams in Figs. 2 to 9, inclusive, shows the following:

1. There is apparently a typical curve, applicable to ores of whatever nature, under like conditions of crushing. This is most clearly presented in Figs. 7 A and 7 B.

2. The curve is merely elevated by successively finer crushing, so long as radial crushing is employed, with no tendency to increase the percentage of fines beyond what would naturally result from the circumstance that the entire 100 pounds of ore is included within narrower limits of size. See Figs. 2 A, 2 B, 4 A, 4 B and 8.

3. There are certain definite sizes to which the ore has a tendency to break, forming prominent culminating points on the curve, irrespective of the method of crushing employed. These culminating points are on No. 2 mesh (11.04 millimeters diameter), on No. 16 mesh (1.06 millimeter diameter), on No. 40 mesh (0.374 millimeter diameter), and between No. 80 mesh (0.171 millimeter diameter) and No. 100 mesh (0.139 millimeter), this last culminating point occurring for the most part on No. 90 mesh (0.155 millimeter diameter). The termination of the curve on the grade called fines, or "slimes," although elevated, cannot be regarded as a culminating point, since there is doubtless a wider difference between the diameters of the coarsest and finest particles in the "slimes" than between the 1 inch and No. 150 mesh material. That there is probably another culminating point on a grade of larger size than 1

Fig. 4B.



Continuation of Fig. 4A, on Reduced Scale. Showing also Effect of Re-Crushing to a Maximum Size, Corresponding to the Size of Mesh on which a Culminating-Point Occurs on the Curve from a Former Crushing. Curves D and E are from Ore Reduced by Radial Crushing.

inch is indicated by curve H, Fig. 6 B, as will appear from a consideration of the following paragraph.

4. Subsequent crushing of a crushed sample, with the crusher set to pass grains of a diameter equal to that of the mesh supporting the size of grains, forming one of the culminating points on the curve, produces in most cases a maximum of grains having a diameter corresponding to the next smaller size of mesh, or else the maximum occurs on the next culminating point. See Figs. 2 A, 2 B, 4 A, 4 B and 8.

FIG. 5.

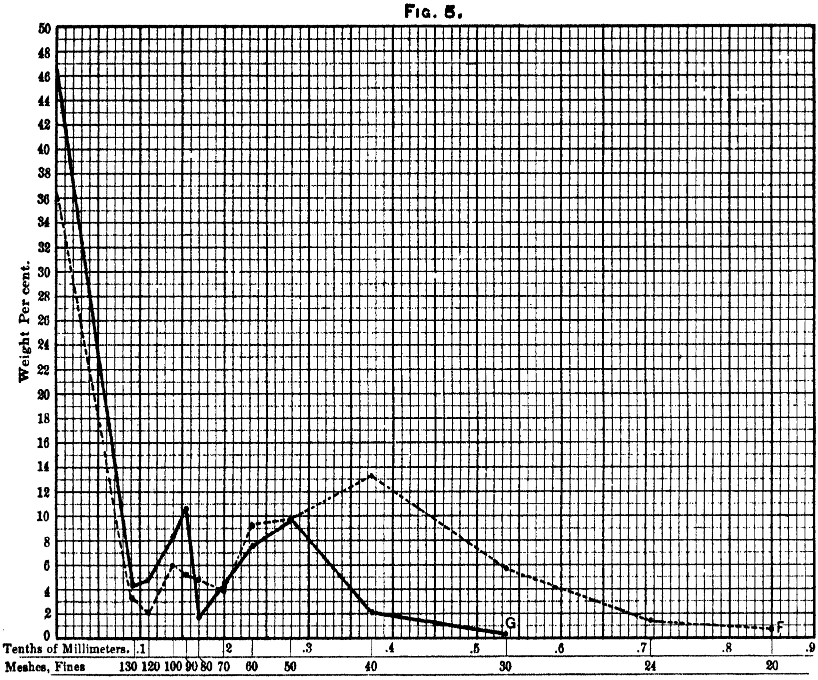


Diagram Illustrating Comparative Results of Shearing and Diametral Crushing. Curve F is from Rock Reduced in Cone-Grinder. Curve G is from Rock Reduced in Stamp-Mill.

5. There is practically little difference in the effects produced by diametral crushing and shearing. See Fig. 5.

It is noteworthy that there is no relation between the ratios of the sizes of particles corresponding to the several culminating points. This may be due to the want of a proper ratio between the successive sizes of the screen openings. Remedying this fault might affect the culminating points, though it could hardly be expected to do more than round the curve at or

about corresponding points. Neither is there any relation which I have been able to discover between the various sizes of products obtained by radial crushing and the volumes of the spherical sectors which were under compression in crushing and the remainders of the spheres which were under tension.

This, however, could hardly be expected when crushing unsized ore. That the various sizes of crushed ore depend upon some law governing the frangibility of rock masses in general would seem to be indicated by the characters of the curves, but further research is needful to ascertain this. It is proposed to

FIG. 6A.

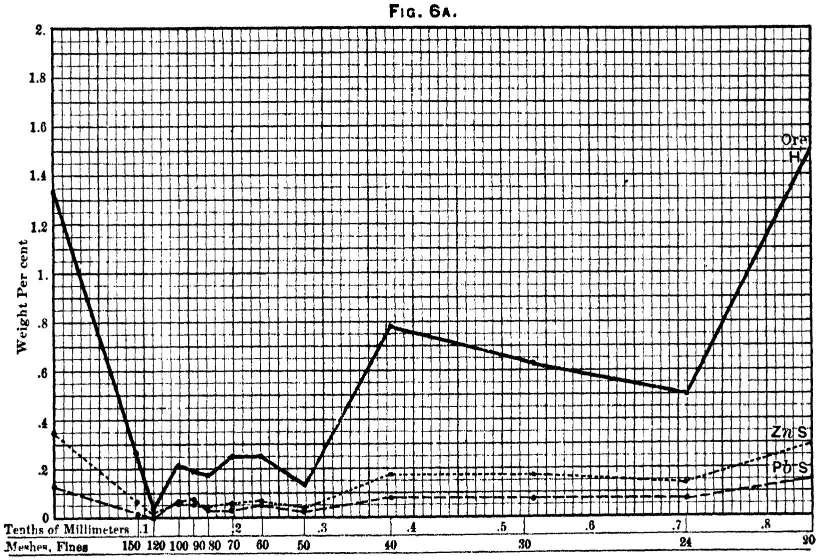
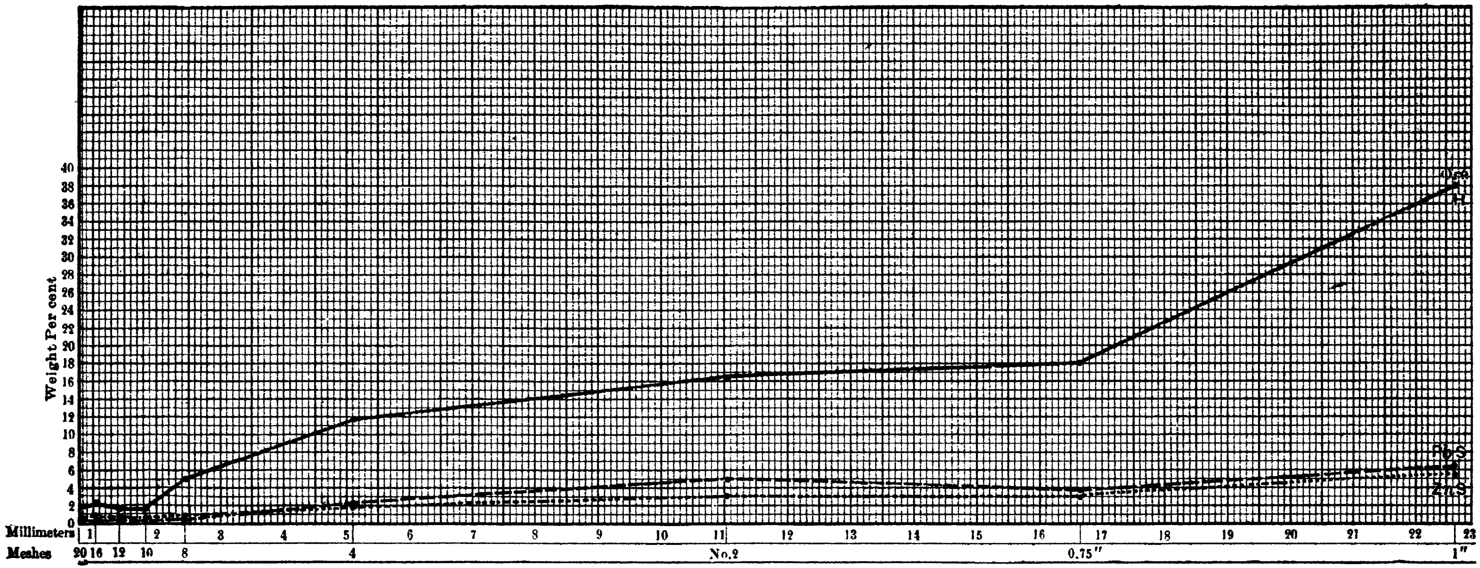


Diagram Illustrating the Distribution of the Relatively Heavy Minerals Through the Various Sizes of Crushed Ore.

continue these investigations by crushing spheres of homogeneous material under conditions in which the crushing-angles and power applied shall be definitely known.

Two examples are given (curves C and H, Figs. 3 and 6 B) in which the distribution of the heavy bodies through the different meshes is also plotted, the quantities on each point of the curves being the percentage of the heavy mineral as referred, not to the quantity of that grade, but to the whole hundred pounds of the original sample. Such investigations are often made, but a sufficiently complete series of sieves is not com-

Fig. 6B.



Continuation of Fig. 6A on Reduced Scale.

monly used to reveal all that could be learned concerning the distribution of the valuable minerals according to size. This is a practical point of great importance in the preliminary study of an ore for concentration. In Fig. 3, for instance, the advantage of screening the rock at No. 12 mesh in order to obtain a rich grade of ore, with a minimum quantity of magnetite, is manifest. The difference in specific gravity between corundum and magnetite is just enough to embarrass one in concentration at all times, increasing the danger of losing corundum in the tailings while seeking as clean a grade of concentrates as possible. In Fig. 6 B we see that the galena has a tendency to produce an excess on No. 2 mesh, which is not shared by the zinc-blende. Crushing to a little less than $\frac{3}{4}$-

FIG. 7A.

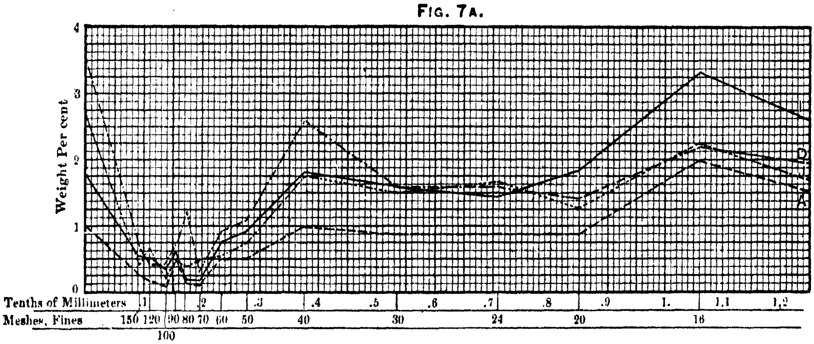
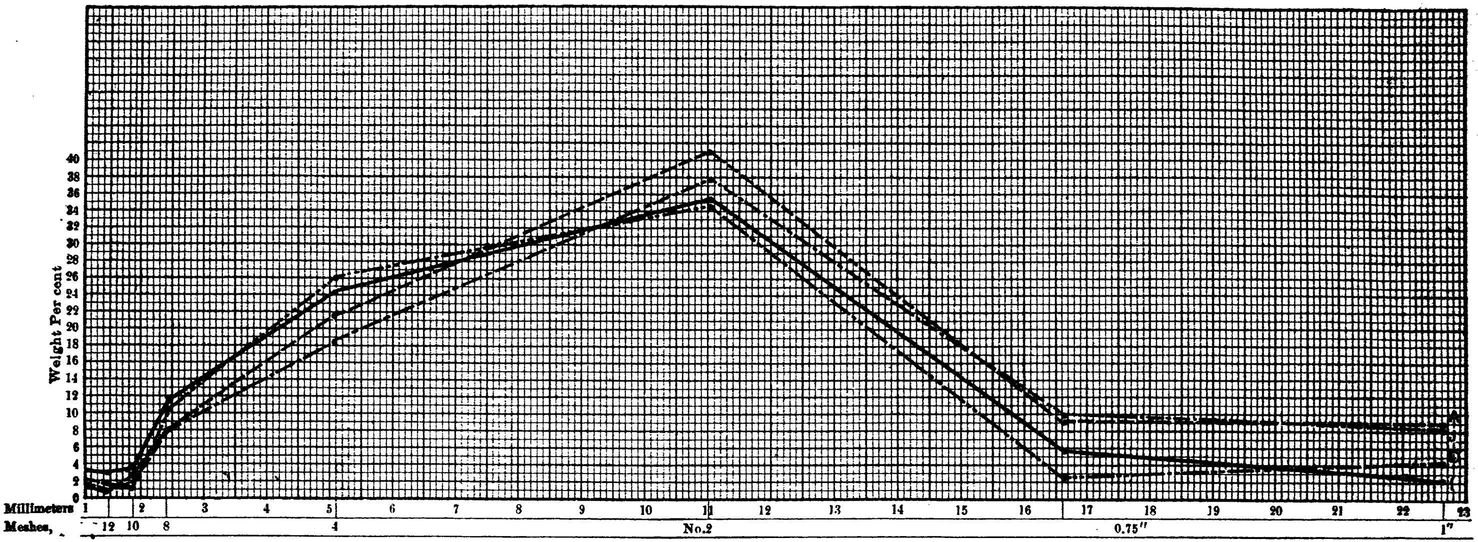


Diagram Illustrating a Marked Tendency to a Typical Curve, Conformed to by Wholly Unlike Ores and Rocks Under Similar Crushing Conditions.

inch size would produce a maximum of Nos. 2 and 4 mesh, in which the relations between the galena and blende would remain approximately as shown. By screening this crushed material through a No. 4 mesh trommel a coarse grade would be obtained having a maximum of galena and a minimum of blende, a most desirable condition for obtaining a clean galena, since the greater portion of the galena grains in this ore broke free from the zinc-blende. Figs. 6 A and 6 B show also that from No. 8 to No. 80 mesh the proportion of blende is in excess of the galena, that the proportions are about equal from No. 80 to No. 120 mesh, while from that size on to finest slimes the proportion of blende is an increasing one—the inferences from which facts, with reference to separating into different grades for concentration, are obvious.

FIG. 7B.



Continuation of Fig. 7A, on Reduced Scale.

Fig. C.

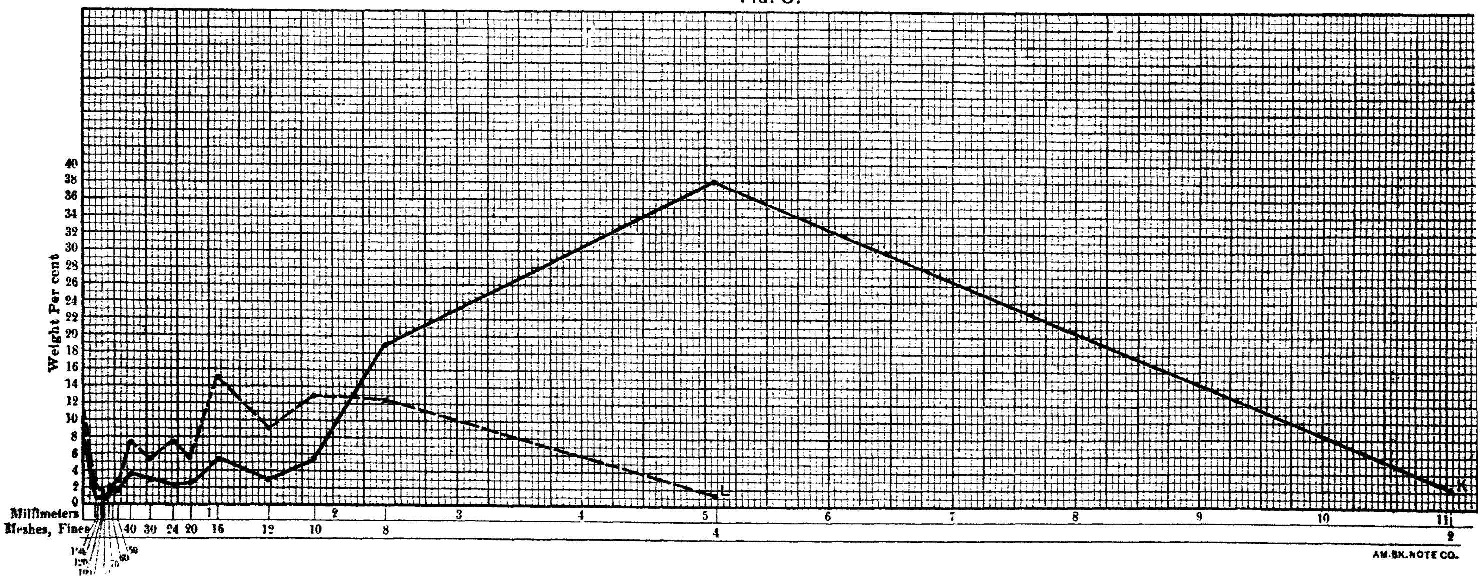


Diagram Illustrating Elevation of Curve, without "Sliming," by Successively Finer Reduction of the Same Ore by Rolls, without Intermediate Elimination of Finer Products. In Curve K the Ore Has Been Reduced Past a Culminating-Point. Re-Crushing to No. 4 Mesh Produces a Maximum of Ore on the Next Culminating-Point, No. 16 Mesh.

FIG. 9A.

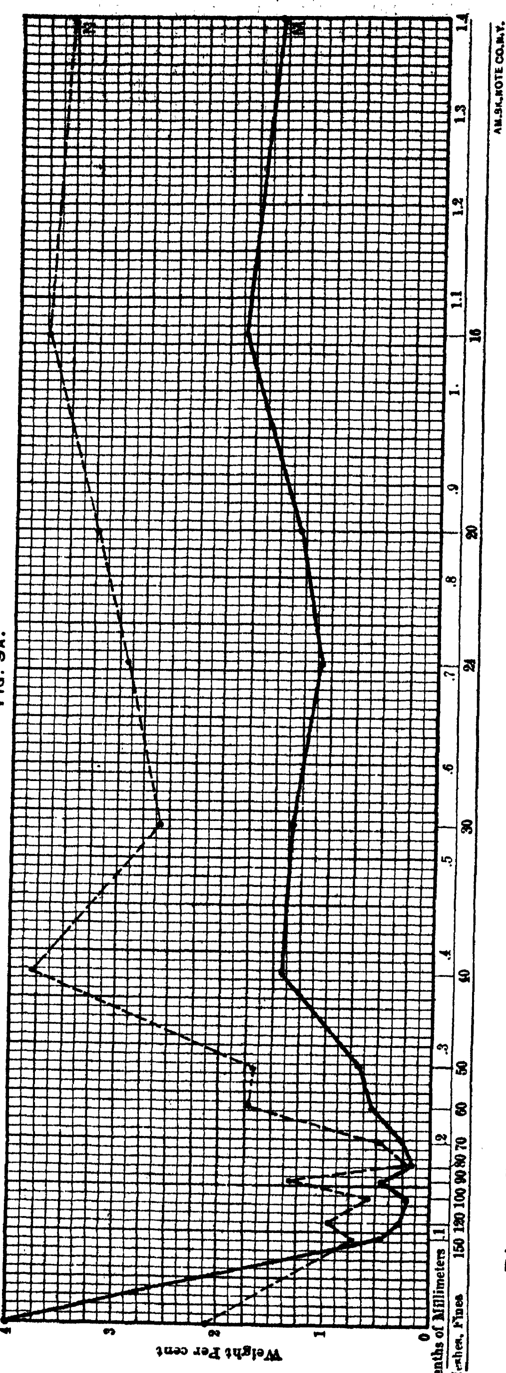
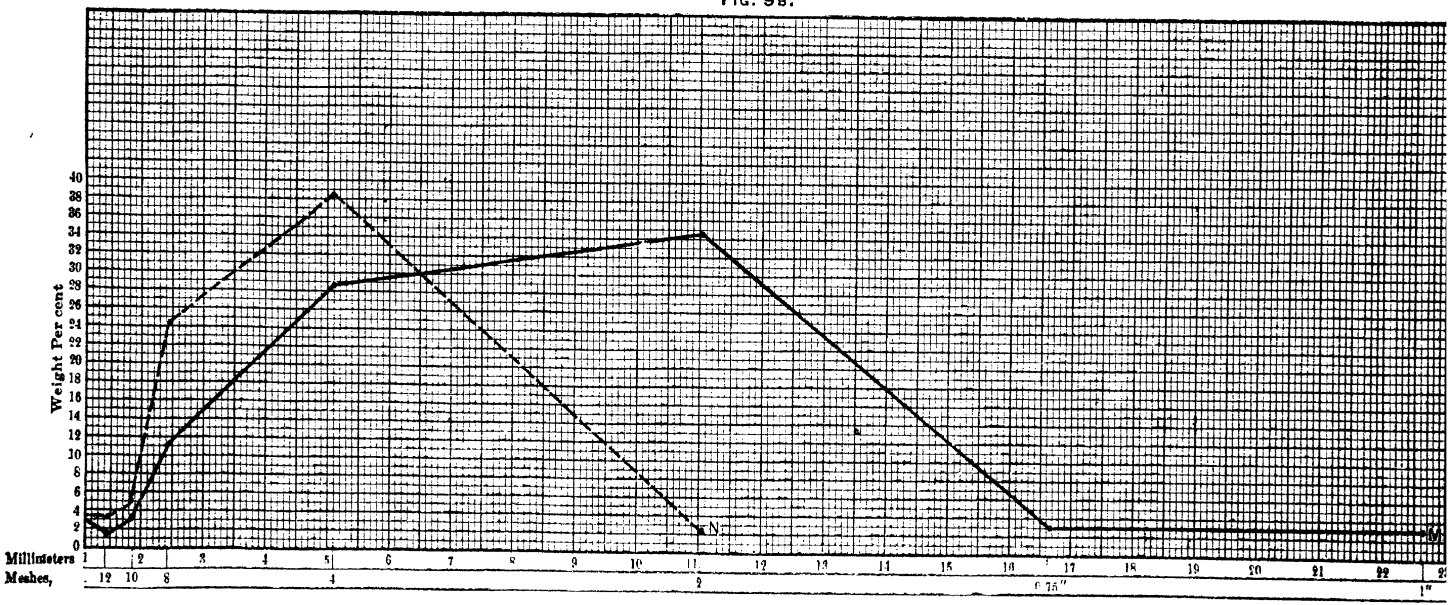


Diagram Showing Relation Between Sizing-Curves of Different Ores, Reduced to Different Sizes by Radial Crushing.

Fig. 9b.



Continuation of Fig. 9A, on Reduced Scale.

TABLE I.—*Results of Screening Different Ores, Crushed to Different Sizes by Various Types of Crushers.*

Sizes.		A.	B.	C.	D.	E.	F	G	H.	I.	J.	K.	L.	M.	N.
Meshes.	Diam. of Openings, Milli-meters.	Corundum Rock, Coarsely Crystalline Syenite. 1″ in Blake Crusher. Per cents.	Same as A. ⅜″ in Rolls. Per cents.	Same as A 0.2″ in Rolls. Per cents.	Calcite (Compact) with 14 per cent. Chalcopyrite. 1″ in Blake Crusher. Per cents.	Same as D. ⅜″ in Rolls. Per cents.	Same as D 0.85 mm in Sample Grinder. Per cents.	Quartz (Crystalline) with Small per cent. Pyrite and 4 dwts. Gold. Stamp-mill. No 30 Screen. Per cents.	Cherty Limestone with Galena and Sphalerite. 1½″ in Dodge Crusher. Per cents.	Pink Dolomite (Crystal Aggregates) with Galena. 1″ in Blake Crusher. Per cents.	Vitreous Quartz with Pyrite. 1″ in Blake Crusher. Per cents.	Vitreous Quartz with Galena. ⅜″ in Rolls. Per cents.	Same as K. 0.1″ in Rolls. Per cents.	Hard, Compact Quartz-schist, with Fine, Disseminated Pyrite. 1″ in Blake Crusher. Per cents.	Corundum Rock. Similar to A, but much less Basic. ⅜″ in Rolls. Per cents.
1 in.	22.61	8.50			4.47				38.000	2.33	8.40			3.10	
0.75 in.	16.62	9.50			2.93				18.000	5.75	10.00			3.08	
No. 2	11.04	41.00	34.50		34.80	24.25			16.500	35.50	37.40	2.20		34.78	2.50
" 4	5.10	21.00	38.80		25.90	36.80			11.333	24.00	18.00	38.00	1.33	28.36	38.36
" 8	2.42	8.00	11.00	18.87	10.30	11.25			4.666	11.25	8.00	19.00	12.19	11.05	24.05
" 10	1.85	2.00	2.62	18.75	1.20	4.03			1.333	3.33	2.75	5.25	13.10	3.00	5.29
" 12	1.48	1.00	1.25	9.31	1.60	2.02			1.166	1.80	1.02	3.00	9.04	1.48	3.50
" 16	1.06	2.00	2.31	14.00	2.20	3.72			2.000	3.33	2.25	5.50	15.09	2.86	3.73
" 20	0.85	0.87	0.87	4.75	1.40	1.45	0.99		1.500	1.85	1.25	2.75	5.55	1.30	3.43
" 24	0.708	0.87	1.00	4.60	1.60	2.02	1.32		0.500	1.45	1.65	2.50	7.54	1.11	2.90
" 30	0.535	0.87	1.06	3.25	1.60	2.13	5.58	0.361	0.625	1.60	1.50	3.00	5.55	1.33	2.61
" 40	0.374	1.00	1.18	5.13	2.60	2.60	13.20	2.004	0.771	1.81	1.75	3.75	7.13	1.43	3.89
" 50	0.279	0.50	0.56	2.60	1.10	2.02	9.90	9.529	0.125	0.90	0.75	1.50	3.23	0.65	1.69
" 60	0.232	0.50	0.60	2.13	0.90	1.01	9.20	7.295	0.250	0.75	0.55	1.50	2.32	0.55	1.72
" 70	0.197	0.10	0.10	0.50	0.30	0.43	4.00	4.896	0.240	0.15	0.50	0.75	0.75	0.25	0.46
" 80	0.171	0.12	0.30	0.60	1.20	0.22	1.63	1.478	0.166	0.20	0.40	0.40	0.58	0.15	0.18
" 90	0.155	0.50	0.30	0.60	0.70	0.79	5.30	10.778	0.188	0.64	0.50	0.95	1.66	0.42	1.33
" 100	0.139	0.10	0.10	0.85	0.40	0.43	6.60	8.282	0.219	0.35	0.20	0.75	1.16	0.20	0.57
" 120	0.110	0.18	0.60	0.87	0.40	0.32	2.00	4.732	0.032	0.54	0.65	0.75	1.40	0.29	0.96
" 150	0.093	0.25	0.60	1.30	0.70	1.25	3.30	4.338	....	0.55	0.40	1.25	2.24	0.45	0.71
Fines.	....	1.00	1.90	6.13	3.50	3.50	36.20	46.268	1.333	1.80	2.65	7.25	10.11	4.03	2.15
		99.86	99.65	94.24	99.80	100.24	99.22	99.961	98.947	99.88	100.57	100.05	99.97	99.87	99.94
		See Figs 2A, 2B, 7A and 7B.	See Figs. 2A and 2B.	See Figs 2A, 2B and 3.	See Figs 4A, 4B, 7A and 7B.	See Figs 4A and 4B.	See Figs. 4 A, 4 B and 5.	See Fig. 5	See Figs 6A and 6B.	See Figs 7A and 7B.	See Figs. 7A and 7B.	See Fig. 8.	See Fig. 8.	See Figs 9A and 9B.	See Figs. 9A and 9B.

TABLE II.—*Distribution of Heavy Minerals Through the Various Sizes of Crushed Ores.*

Sizes.		Sample H. Cherty Limestone, with Galena and Sphalerite, Joplin, Mo. Crushed in Dodge Crusher, set to 1½ inch.			Sample C. Corundum Rock (Coarsely Crystalline Syenite) Crushed in Blake Crusher, set to 1 inch, and then Twice in Rolls to 0.2 inch.			
Meshes.	Diam. of Openings. Millimeters.	Ore · Per cent. Remaining on each Screen.	Pb S · Per cent of Total Ore on each Screen.	Zn S Per cent of Total Ore on each Screen.	Rock Per cent. Remaining on each Screen.	Specific Gravity.	Corundum: Calculated per cent of Total Rock on each Screen.	Magnetite: Per cent of Total Rock on each Screen.
1 inch.	22.61	38.000	6.69	5.48				
0.75 in.	16.62	18.000	3.86	2.89				
No. 2	11.04	16.500	5.08	3.15				
" 4	5.10	11.333	2.27	2.18				
" 8	2.42	4.666	0.63	0.86	18.87	3.08	8.12	0.38
" 10	1.85	1.333	0.15	0.26	18.75	2.88	4.68	0.56
" 12	1.48	1.166	0.12	0.27	9.31	2.91	2.60	0.28
" 16	1.06	2.000	0.33	0.47	14.00	2.91	3.35	0.98
" 20	0.85	1.500	0.15	0.29	4.75	2.95	1.31	0.30
" 24	0.708	0.500	0.06	0.14	4.60	2.92	1.23	0.32
" 30	0.535	0.625	0.07	0.17	3.25	2.93	0.80	0.25
" 40	0.374	0.771	0.08	0.17	5.13	2.97	1.47	0.39
" 50	0.279	0.125	0.02	0.03	2.60	2.94	0.53	0.27
" 60	0.232	0.250	0.05	0.06	2.13	2.96	0.54	0.20
" 70	0.197	0.240	0.02	0.05	0.50	3.00	0.15	0.05
" 80	0.171	0.166	0.03	0.04	0.60	2.94	0.15	0.06
" 90	0.155	0.188	0.07	0.05	0.60	3.04	0.20	0.06
" 100	0.139	0.219	0.06	0.06	0.85	2.985	0.24	0.08
" 120	0.110	0.032	0.002	0.008	0.87	3.07	0.30	0.09
" 150	0.093	......	......	.....	1.30	3.03	0.42	0.10
Fines.	......	1.333	0.13	0.35	6.13	2.96	1.85	0.27
Totals.		98.947	19.872	16.978	94.24		27.94	4.64
		See Figs. 6A and 6B.			See Figs. 2A, 2B and 3.			

The diameters of openings in the sieves used in these experiments are given in Tables I. and II. The sieves were all of the best crimped brass wire. It is unfortunate that the diameters of openings corresponding to different meshes vary with different makers of screen cloth. Although the change would be somewhat radical, it would be a distinct advantage if the metric system could be substituted for the old inch-system, the sieve-openings bearing definite relations to each other

through the different sizes, and the diameter of wire always being the same for the same number of openings to the centimeter. Without such uniformity, comparison of results is rendered extremely difficult.

The Effect of Sizing on the Removal of Sulphur from Coal by Washing.

BY CHARLES C. UPHAM, NEW YORK CITY.

(Buffalo Meeting, October, 1898.)

NOT long ago a few acres of coal-land in the Connellsville region of Pennsylvania were sold at the rate of $1500 per acre. While this was doubtless a "fancy" price, affected by some consideration other than that of the simple value of the coal in the tract, it may fairly be taken as an indication of a progressive and already perceptible diminution in the economically available coal-resources of that famous region. This present and prospective development emphasizes the commercial importance of producing, from the inferior coals of other districts, coke suitable for use in the blast-furnace.

Many of the bituminous coals of the United States would meet this requirement if sufficiently freed, before coking, from slate and sulphur, and, so far as the removal of slate is concerned, the problem is not specially difficult. There are many forms of trough-, jig- and gravity-washers which easily and cheaply reduce the slate from as much as 20 per cent. in the coal to as little as 8 per cent. in the coke. But to reduce the sulphur from 4 per cent. in the coal (a proportion sometimes encountered) to 1 per cent. or less in the coke (the proportion demanded by the furnace-manager) is not so easy.

It may be safely assumed as familiarly known to members of the Institute interested in this subject that, of the three forms in which sulphur usually exists in coal (hydrogen sulphide, calcium sulphate and pyrite), the pyrite is the one to be removed, as far as possible, by mechanical treatment before coking. Hydrogen sulphide, being volatile, is expelled in

coking; calcium sulphate, or gypsum, is not practically removable by preliminary treatment, and remains in the coke, while pyrites (the most abundant source of sulphur in coal) may be, to some extent, oxidized, with removal of its sulphur, in the coke-oven, but can be, by reason of its high specific gravity, more advantageously removed by mechanical treatment before coking.

Concerning the removal of sulphur in the oven, it may be observed, in passing, that this result (at best, only partial) seems to depend upon something more than the simple oxidation of the sulphide. The presence of water—that is, of hydrogen as well as oxygen—apparently promotes the expulsion of sulphur in a volatile form. It has been observed that coal charged wet into beehive ovens will yield a coke containing less sulphur than if charged dry, and the same is probably true of closed retort-ovens; but in the latter case, no doubt, experience would prove, as it has done in the former case, that wet coal requires a longer time in coking.

It is therefore evident that the pyrites should be removed, as far as practicable, before the coal is charged into the coke-oven, and it is the purpose of this paper to call attention to an element in this preliminary treatment which is, according to my observation, very generally neglected in the washing of coal in this country.

It may be proper to say here that this point was forced upon my attention in connection with certain experiments on a large scale (which need not be more particularly described at present) in which the presence of pyrites in coal was found to be disadvantageous. I was, consequently, led to study the general practice in the washing of coal, and to make certain experiments, described below.

When coke-ovens are located at the mines the slack produced in the ordinary screening of the coal for market is very commonly washed and coked. But in many cases the coal is treated just as it comes from the screens—in pieces varying all the way from an inch-cube down to dust—while in few, if any, cases is the slack separated into more than two or three sizes. Of course, in the process of separation by specific gravity, pieces of coal or slate with adhering pyrites, having an average specific gravity, perhaps midway between coal and slate, or

even very nearly that of the clean coal, may go to the coal-side of the apparatus; but a preliminary crushing and sizing, reducing all pieces below a given maximum diameter (to be determined, doubtless, for each coal separately, but to be probably not more than ¼-inch diameter for any coal requiring washing) will separate all the pyrites from the coal or slate, leaving each to the action, in the process of washing, of its own specific gravity. Experimental illustration of this proposition is given below.

Again, a portion of the pyrites in coal as it comes from the mines seems to be in particles finer even than those of the coal-dust and constituting an impalpable powder. This "flour-pyrites" floats in air or water. The following experiments were made to test its relative fineness:

A sample of finely-crushed coal was passed over a 20-mesh screen. The material remaining on the screen contained, by analysis, 1.11 per cent., while that which passed through contained 1.49 per cent. of sulphur.

Again, a sample of coal-dust was submitted to suction from a centrifugal blower. The part thus taken up and deposited in a separate chamber contained 1.36 per cent., while the remaining portion, not acted upon by the suction, contained 1.14 per cent. of sulphur.

These results, especially in view of the higher specific gravity of the pyrites, seem to prove conclusively that it pulverizes more finely than the coal.

The practical importance of this fact will be evident, when it is considered that, in many plants where coal-slack is washed for coking, the washed coal is passed over screens and the water is drained off, to be returned to the washing-apparatus and used again. Obviously, this practice cannot give the best results; for on the one hand, some of the "flour-pyrites" will be left on the coal in draining, and some will be carried back by the water, to accumulate by successive operations; and ultimately pass off with the coal.

With regard to both the foregoing suggestions, but chiefly as to the importance of the finer crushing of coal before washing, the following experiments may be of interest:

They were made upon coal from the Pittsburgh seam, and on a working scale, with samples of several tons. The appa-

ratus employed was a trough-washer, of a type in general use. Since the same apparatus was used in all the experiments, its special merits may be regarded as not requiring attention in connection with this particular investigation.

A sample, A, of ordinary slack, and a sample, B, of the same slack, crushed to particles $\frac{3}{16}$-inch and less in diameter, were treated in the washer and the cleaned material was analyzed. The analyses of the original slack and of the result from each sample, were as follows:

	Ash. Per cent.	Sulphur. Per cent.
Unwashed slack,	10.51	2.876
A, washed,	7.97	2.230
B, "	4.56	1.188

The result from sample B would make coke carrying about 1 per cent. of sulphur.

Another sample of Pittsburgh coal, containing 11.95 per cent. of ash and 2.121 per cent. of sulphur, was crushed to about 20-mesh size, and washed as before, the result showing 4.86 per cent. of ash and 1.046 per cent. of sulphur. The coke made in a crucible from this sample contained 0.836 per cent. of sulphur—an amount well within the commercial limit.

In this experiment, the coal and water, after passing through the separator, were led into a settling-bin, where the coal was deposited at the bottom, while the water (and with it the "float-pyrites") was conducted away from the top in a stream half an inch deep. So much of the fine pyrites was carried on the surface of the water that the glistening particles could be readily skimmed off with the hand.

Additional experiments made with other coals have shown that the critical size at which an almost complete division of the coal and pyrites takes place varies with coals from different districts and beds. In designing plants for coal-washing, the proper fineness of crushing should be determined beforehand by careful experiment.

As a result of the observations and experiments above described, I have been led to believe that, by due observance of the conditions indicated, coke of good quality could be made from many coals not now considered as suitable for that purpose, and many inferior cokes now in the market could be greatly improved.

The Alluvial Deposits of Western Australia.

BY T. A. RICKARD, STATE GEOLOGIST, DENVER, COLORADO.

(Buffalo Meeting, October, 1898.)

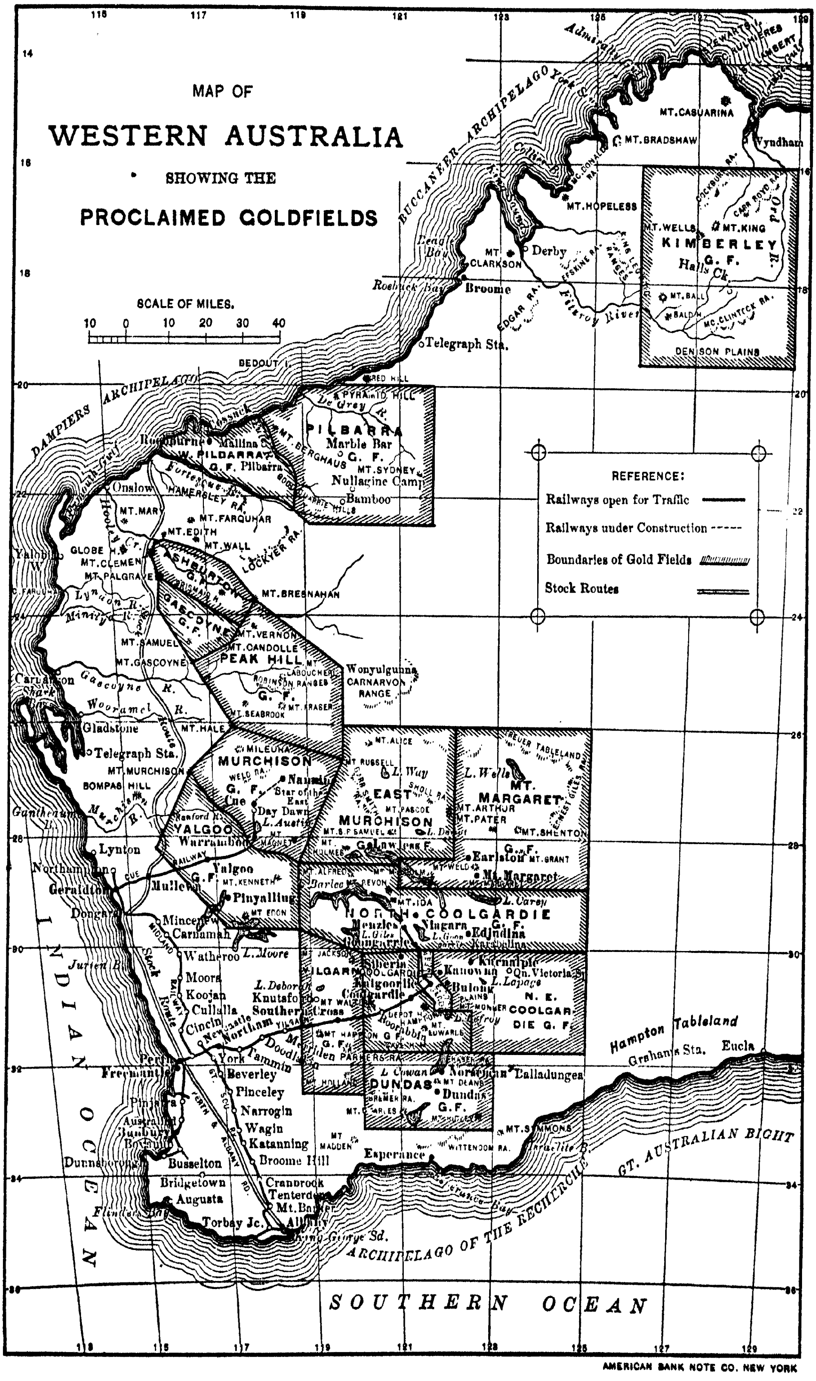
I.—General Geological Considerations.

The interior of West Australia is an arid table-land, elevated 1400 feet above the sea. This plateau is flanked to the south by the Tertiary limestones which fringe the Great Australian Bight. It is bordered northward by the Carboniferous beds of the Fitzroy river and westward by the granite of the Darling hills, while to the east this wide area, about 900 miles square, slopes downward imperceptibly into an undulating plain of sand, which stretches with dismal persistence across the boundary of South Australia. The waters of the ocean receded from this tract of land long ago; it is probably the oldest land-surface on the globe, and represents the basal wreck of a much larger continent. Fig. 1 is a map of this region.

The Coolgardie and Kalgoorlie gold-fields are situated in the southwestern part of the region. The rock-formation consists of granite penetrated by diorites and andesites. The latter are occasionally associated with tuffs, which have been readily mistaken for sedimentaries. There are no fossil-bearing rocks, such as would afford a datum-line from which to measure the relative geological age of the prevailing formation. On the extreme edges of the mining territory there are, it is true, remnants of sand-rock which are considered identical with the "Desert Sandstone" of Queensland, determined by Daintree to be of Mesozoic age. But even this formation has evidently been laid down so long subsequent to the underlying rocks that it serves merely to emphasize their much greater antiquity.

In their characteristics and in their relations to each other, the granite and the diorite of the Coolgardie region appear to me much to resemble the Laurentian granite and the Huronian

Fig. 1.


MAP OF
WESTERN AUSTRALIA
SHOWING THE
PROCLAIMED GOLDFIELDS
SCALE OF MILES.
10 0 10 20 30 40
REFERENCE:
Railways open for Traffic
Railways under Construction
Boundaries of Gold Fields
Stock Routes
KIMBERLEY G. F.
PILBARRA G. F.
W. PILBARRA G. F.
ASHBURTON G. F.
GASCOYNE G. F.
PEAK HILL G. F.
MURCHISON G. F.
EAST MURCHISON
MT. MARGARET G. F.
YALGOO G. F.
NORTH COOLGARDIE G. F.
N. E. COOLGARDIE G. F.
DUNDAS G. F.
INDIAN OCEAN
SOUTHERN OCEAN
GT. AUSTRALIAN BIGHT
Hampton Tableland
AMERICAN BANK NOTE CO. NEW YORK

schists of Ontario, in Canada.* Their age can only be vaguely described as Archæan.

On many parts of the earth's surface a long-continued, slow movement of continental uplift, interrupted by intermittent periods of rest or subsidence, has permitted the transfer of land to the sea by the erosion of the exposed parts and the deposition of their detritus as ocean sediment, thus causing the upbuilding of a mountain-system composed of a series of rocks belonging to successive epochs. In this particular region, on the contrary, all diversity is wanting, and a sameness of aspect wearies the observer. In the absence of an elevatory movement, more than sufficient to balance the slow degradation of the higher parts of the region, there has been no compensation for the effects of atmospheric erosion, so that this tract has become a dreary flat, strewn with the sandy wreck of weathered rocks.

The United States offers both a contrast and a parallel. In the Rocky Mountain region the movement of uplift which commenced in pre-Cambrian times has only been interrupted so as to permit of the laying-down of younger sediments; and the degradation of the high places has been compensated, and sometimes exceeded, by an elevation which has resulted in the formation of a mountain mass flanked by a long succession of strata now enclosing a great variety of mineral wealth. The interior of Australia can be likened to the Great Basin, occupied by Nevada and parts of Utah, Idaho and Arizona, between the Rocky Mountains themselves and the Sierra Nevada. There is only one large river in Australia which reaches the sea, namely, the Murray, which rises near the boundary-line of Victoria and New South Wales, and then flows toward the interior, to be saved by a backward sweeping curve, which permits the river at length to empty itself into the sea at the border of South Australia. There are many "lost rivers," like the Carson

* The mass of the granite is penetrated by the mass of the diorite, the latter being therefore the younger; but puzzling evidence is afforded by the fact that the diorite at the contact is sometimes traversed by embranchments of granite, which are explainable on the supposition that subsequent metamorphism gave the granite a renewed mobility, permitting it to penetrate fractures in the diorite. At Rat Portage, Ontario, the overlying, younger Huronian schists are intercalated and penetrated by the older Laurentian granite at certain places along the line of contact.

and the Humboldt in Nevada. During the rainy season they are tempestuous torrents; during the succeeding dry months their course is marked by sandy bottoms, dotted with an occasional water-hole. The mountains are near the coast, so that the Australian Alps and the Blue Mountains do the same service as the Sierra Nevada and the Cascades, in that they interrupt the warm, moisture-laden winds, and compel them to precipitate on the seaward slope. The consequence is that the eastern parts of the colonies of Queensland, New South Wales and Victoria, between the mountains and the sea, resemble the valleys of California and Oregon, just as the interior region beyond them bears a likeness to the dry tracts of Nevada and Arizona.

The sea retired from the interior of West Australia in the very dawn of geological time, and the movement of elevation, which raised the land above the waters, continued with but little interruption until the beginning of the Tertiary period. Since then, slow subsidence has robbed the Australian continent of a part of its extent, and made Tasmania an island. There is evidence of a much larger continental area, which at one time extended toward Southern Africa. The encroachment of the sea has crowded a wonderful variety of flora into the small stretch of fertile country lying between the desert and the Indian Ocean.*

The main drainage of the interior is to the south. The last retreat of the sea was accompanied by the formation of broad valleys, which have lost their former outlines, and now appear as long depressions, largely filled up with the products of erosion.

II.—The Physiography of the Gold-Fields.

The principal gold-field of West Australia is situated in the southwestern part of the desert plateau. The chief towns are

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* This corner of Australia is celebrated among botanists for the extraordinary variety of its flora. Baron Ferdinand von Mueller, the celebrated botanist, may be quoted: "A marvellous exuberance of plants, different in species, and often gay or odd in aspect, exists within a triangle formed by a line of demarkation drawn from the south of Sharks Bay to the west of the Great Bight; and within this space are chiefly located those species which are exclusively restricted to West Australian territory."

Coolgardie and Kalgoorlie.* They are connected by 350 miles of railway with the coast. During the year 1897 the total rainfall amounted to 5⅝ inches at the one place and 4¾ inches at the other.† In contrast to these figures, it may be added that the rate of evaporation in this region is estimated to be equivalent to 7 feet per annum.

The country consists of a sandy plain, the monotony of which is intensified by a series of alternating low rocky ridges and equally slight depressions, having a northwesterly direction. Most maps indicate the occurrence of lakes and the occasional course of a stream, but these are the mirages of the cartographer. The "lakes" are shallow basins with clay bottoms, in which, during the rainy season, a little water lingers, and the "streams" are sandy channels, where sinking will sometimes tap a trickling flow of brine.

The surface is devoid of vegetation, except in spring, when flowers‡ of a brilliant hue, but with the texture of hay, leap into brief existence. Animal life is infrequent. An occasional bustard may be provoked into leisurely flight, a troop of paroquets throws a momentary gleam athwart the dull gray of the bush, or a solitary kangaroo hops across the trail. These, however, are but infrequent interruptions to the sullen silence of the wilderness.

The real nakedness of the region is hidden by the "bush," consisting of scrub from 20 to 60 feet high, chiefly mulga and ti-tree.§ This covers all things as with a garment (see Fig. 2). The roads are cut through it with the monotonous regularity of a canal. One portion of the journey is but the counterpart of another. The sameness is wearisome beyond words. And

* These localities are not found save on recent maps. Kalgoorlie is in latitude 30° 45′ south and longitude 121° 30′ east, while Coolgardie lies in latitude 30° 57′ south and longitude 121° 10′ east. They are 25 miles apart.

† The rainfall at Denver is 14½; Alexandria, 10; Paris, 22; London, 35; Canton, 39; Calcutta, 76; Vera Cruz, 180; and at Cherrapongee, in Assam, 610 inches per annum.

‡ The "Everlastings," as they are usually called, belong chiefly to the genera Helichrysum, Helipterum, Waitzia, Podolepis and Angianthus. For about three months they appear as magnificent splashes of color, carpeting the desert with splendor. They are wholly devoid of perfume, and have the brittle texture of artificial flowers.

§ Both acacias. The characteristic tree-shrubs of the country belong to the genus Acacia. Many of them have a fragrant bloom in the spring.

when an elevated spot is attained the eye commands, from north to south, from east to west, one dark unbroken sea of trackless bush.

Gold-mining caused this desert to be invaded. The first discovery was made by Anstey, in 1887, at Yilgarn, which is 210 miles east from Perth, the capital of the colony. This started the Southern Cross mining district. Prospectors began to scatter further inland. In 1892 Bayley made the discovery which marked the birth of Coolgardie, and the commencement of an activity which culminated in the mining excitement of 1895. A series of rich finds, scattered over the surrounding desert, gave rise to the settlements of Menzies, Goongarrie, Kanowna, Kurnalpi, Kunanalling, Wagiemoola, and a score of other patches of corrugated-iron hideousness labeled with euphonious aboriginal names. In June, 1893, Patrick Hannan pegged out a discovery-claim at Kalgoorlie,* 25 miles east of Coolgardie. The find which he made was of no particular importance, and the neighboring area, like many others, became the scene of the purposeless digging which was at that time sufficient to give an impetus to a great deal of reckless company-promoting. However, just as, in Colorado, the Mt. Pisgah fiasco of 1884 preceded the real development, eight years later, of Cripple Creek, so the vagaries of irresponsible schemers led to the accidental opening up and the eventual recognition of the magnificent series of rich lodes that have now placed Kalgoorlie among the few great mining camps of the globe.

In 1897 West Australia produced 674,993 ounces of gold, to which Kalgoorlie alone contributed 306,000 ounces. During the same period the mines of the colony paid $2,400,420 in dividends, and out of this total Kalgoorlie is credited with $1,775,000. The growth of the industry is exhibited by the accompanying statistics:

Year.	Kalgoorlie. Ounces.	West Australia. Ounces.
1895,	36,000	231,513
1896,	103,000	281,265
1897,	306,000	674,993

It is estimated that the yield of the colony for the current

* See the interesting paper of my friend, Mr. George J. Bancroft, "Kalgoorlie, Western Australia, and Its Surroundings," read at the Atlantic City Meeting, February, 1898, and printed in the present volume, page 88.

year will reach a million ounces, and that of this one-half will come from the Kalgoorlie district.

The development of a group of very rich telluride lodes amid this immense desert country, dotted over with the unhappy failures which were based on small pockets of specimen gold-quartz, did not happen without a sad expenditure of money and human life. With the whisper of every new discovery, crowds of reckless gold-seekers plunged madly into the outer desolation. Eager horsemen jostled the awkward camels, whose swinging gait carried them in turn past the mobs of diggers who trudged wearily to the scene of each successive excitement. One knows not whether to admire the pluck or to deride the foolishness of men who died of thirst and perished of fever in the mad search for gold. The incident known as "the Siberia rush" will be typical of early days. A man came into Coolgardie one night with a story that gold had been found at a locality thirty-odd miles to the north. Hundreds started off on horses or on camels; many went on foot, carrying their billies* and blankets upon their shoulders or trundling their packs in wheelbarrows. Some took the wrong direction, and of these many never reached their destination, but died miserably in the bush. Four hundred eventually reached Siberia.† The only water near the discovery was a soak,‡ seven miles distant. It was soon drained dry by the crowd of diggers. News came to Coolgardie that a water-famine was imminent. The superintendent of water-works, a government officer, instantly despatched a dozen camels§ to the succor of the adventurers. In the meantime, they, realizing the impending danger, had left the gold, and were making for the nearest condenser.|| Many died on that return journey, and many more would have been lost save for the water brought by the camel-train. But in a few days there was another stampede in another direction. Thus the gold-fields were opened up. *Sic Etruria crevit.*

* The "billy" is the tinned-iron vessel, of from 2- to 4-quarts capacity, in which the miner makes his tea and does his cooking.

† What a satire is the name! The locality has a mean annual temperature of 78°, and the summer heat is 112° to 120° in the shade.

‡ A "soak" is the morass of the desert, where water has accumulated in a depression, and is got by digging through the sand which covers it.

§ A camel carries two tins, each holding 20 gallons.

|| All the drinking-water is the product of the "condenser," of which a description is given below. See Figs. 3, 4, 24, and 25.

The peculiar character of these "rushes" is directly traceable to the nature of the gold-occurrence. Gold is found lying on the very top of the ground, and the first surface-mining yields extraordinary profits. The search for the particles of gold scattered over the surface is called "specking." In the early days hundreds of ounces were thus picked up in a few hours by the men first to reach a rich spot. When the cream has thus been skimmed off, the sandy soil underneath is treated, the dirt being winnowed by pouring it from one pan into another. After that, actual digging begins, the shallow deposits being trenched and pitted in the search for those patches of rich ground through which the gold is found sporadically distributed. "Specking" is still a recognized occupation on Sundays,* even at the established mining centers. I have seen as many as a hundred men walking about with their hands in their pockets and their eyes intent on the ground, for all the world as if they were in disgrace. A five-ounce nugget may be found; and everyone hastens to the spot. Perhaps nothing more is picked up; or it may be that sufficient gold is discovered to attract troops of "dry-blowers" to the place. The "dry-blower" is brother to the "gulch-miner" of America and the "alluvial digger" of the eastern colonies of Australia. In the investigation of the methods of the dry-blower and the occurrence of the deposits out of which he wins the gold, I observed many facts of such interest, it seemed to me, as to warrant the preparation of this contribution to the *Transactions* of our Institute.

In mountainous regions the disintegration of the surface is mainly caused by the frost. Water penetrates into crevices and cracks, and, because its maximum density is at 4° C. and not at zero, it undergoes such expansion in the immediate approach to the freezing-point as to become a powerful lever for tearing the rocks apart. Thus is loosened that material out of which eventually the alluvial deposits of the valley are formed. In warmer climates the contraction and expansion of water is likewise a ceaselessly destructive agent. Even in a dry, hot region, such as the interior of West Australia, the difference

* Sunday labor is generally forbidden throughout the Australian colonies; hence the opportunity to go "specking," as above described.

between the heat of day and the cold of night causes the dew to play an important part in moulding the physiography of the country. For it must be remembered that the changes of volume caused, in water as in other substances, by changes of temperature, are well-nigh irresistible in energy, however minute in amount. The cool nights, which alone make life bearable on the Coolgardie gold-fields, are thus beneficent in two ways. To them is ultimately owing the formation of those accumulations of gold-bearing dirt out of which many a prospector has dug himself a competence for life. The tables on page 499 from the government meteorological reports exhibit this variation in temperature.

In mountainous lands the melting of the winter snows yields the water employed in that process of concentration which begins as soon as the rock is shattered, and continues until each ultimate particle has been classified by the untiring machinery of nature. The "tailings" are the mud and sand which go to build new continents upon the ocean-floor, the "middlings" are the great masses of alluvium covering the plain, and the "heads" are the gold-bearing gravels of the mountain-valley. The soft is separated from the hard, the heavy from the light, until at length the metal once incased in quartz and enclosed within the rock is set free, to be collected wherever the eager stream has so abated its force as to permit the particles of gold to find a quiet resting-place.

It is a great sifting-process. The motion of the water is governed by the slope of the surface. In a flat country the conditions resemble those surrounding a mill so situated as to be incapable of getting rid of its accumulating tailings. Should the water-supply of the mill also prove insufficient to carry out its operations, then the analogy with a desert plateau is complete. The process of concentration must in both cases remain unfinished.

The gold-bearing gravels of California and Victoria, for example, usually rest on a hard bed-rock, whose water-worn surface speaks of the agency which made it so. The particles of gold and heavy iron-sand have been washed clean; the overlying pebbles and quartz gravel are comparatively free from material less resisting than themselves; and, if there be any clay, it is found in distinct layers, in positions testifying to the varia-

TABLE I.—*Meteorological Conditions at Coolgardie*, 1897.

Month.	Temperature.					Temperature of Dew-point.		Rainfall.	
	Mean Max.	Mean Min.	Highest Max.	Lowest Min.	Greatest Variation in One Day	Mean. 9 A.M.	Mean. 3 P.M.	Total Inches.	Days.
	°	°	°	°	°	°	°		
January	94.1	63.2	104.3	53.0	42.2	53.5	54.0	.56	4
February ..	89.4	58.5	104.6	47.4	47.8	51.8	53.6	.54	5
March.	85.6	57.7	98.4	50.0	41.0	49.2	55 0	.10	2
April	81.2	53.9	96.1	39.1	36.5	51.3	57.4	.01	1
May...........	71.4	46.6	88.4	38.2	41.5	46.9	53.5	.09	8
June	62.9	43 9	71.2	31.5	29.8	44.2	49.7	1.04	9
July...........	65.1	42.4	74.0	36.5	33.5	43.5	47.5	.34	6
August	63.8	41.5	81.0	33.0	34.6	40.8	43.7	1.08	10
September..	75.0	47.5	92.0	35.0	39.7	43.7	49.5	.29	6
October.. ...	81.0	49.8	91.0	41 0	39.7	......	......	06	2
November..	90.6	58.4	105.0	47.3	44.6	......	. .	.09	1
December..	91.7	59.5	109.2	51.0	44.2	......	......	1.31	4
					1898.				
January	97.6	65.1	111.2	54.0	43.2	60.1	68.2	Nil.	...
February .	89.3	62 5	107.2	48.0	37.7	57.0	60.2	.27	1

TABLE II.—*Meteorological Conditions at Kalgoorlie*, 1897.

Month.	Temperature.					Temperature of Dew-point		Rainfall.	
	Mean Max	Mean Min.	Highest Max.	Lowest Min.	Greatest Variation in One Day.	Mean. 9 A.M.	Mean. 3 P.M.	Total Inches.	Days
	°	°	°	°	°	°	°		
January...	92.9	66.0	105.0	55 0	34.2	51.1	52.0	.38	2
February	88.2	61.6	103.0	49 0	34.7	51.2	51 5	.02	1
March.	83.8	58.9	98.4	51.0	39.0	49.0	49 4	.52	6
April	80 0	55.9	95.4	38.8	32.8	48.0	46 8	20	1
May	69.8	48.0	88.1	37.0	40 4	46.5	45 0	.10	1
June.....	62 9	47.4	73.2	36.2	30.2	46.4	49.1	1 26	14
July	64.4	43 3	74.0	33.2	29.8	42.8	43.7	.22	3
August. .	63 8	43.3	82.0	34.0	29.1	40.0	41 7	.65	9
September.	74.2	48.8	90.8	37.2	40.2	41 6	43.8	.41	5
October. . ..	78.5	51.8	90.2	41 0	38.0	42.0	40 4	.11	2
November	90.2	59 0	103.0	48.0	49.0	46.8	48.7	.06	1
December..	90.7	61.6	109.2	49.4	39.0	48.3	49.5	.82	4
					1898.				
January..	98 0	66 8	113 2	55.0	41.0	51 3	50 3	.02	1
February..	90 5	63.4	109.8	48.2	38.6	51.2	49 9	.36	3

tions in the velocity and volume of the stream which laid down the deposit as a whole. It is an orderly arrangement of assorted material.

Compare with this the alluvium of the desert. A low ridge is crested with the outcrop of a gold-bearing quartz-vein which, amid that surrounding sea of dark-blue scrub, justifies its colonial designation, a "reef." On its flanks there is a thin cover of

sandy soil which gradually thickens, a little lower down the slope, to a deposit of several feet. Sink a hole and you will find, first an inch or two of loose sand and dust, then a more solid layer of gravel and dirt, which in turn passes imperceptibly into a compact material consisting of fragments of rock and quartz, held firmly together by clay. It is called "cement," and it might better be termed an "agglomerate." It is an unclassified product of erosion, and lies close to the place of its origin, as a mere collection of unsorted *débris*.

The rock on which the deposit rests is so softened by decomposition that it is frequently taken for a part of the overlying detritus. If the hole be continued so as to become a well or shaft, it will penetrate through additional oxidized ground until this suddenly gives place to diorite or granite (the two prevailing formations) at a depth varying from 75 to 200 feet, which is the drainage-level of the region.

This deposit owes its existence to the wind and rain, assisted by gravity acting on a slightly inclined surface. Wind is ordinarily an insignificant geological agent, but in the constant and violent draughts of a high plateau there is a force which, working during long periods of time, is capable of producing notable results. In the vicinity of Coolgardie and Kalgoorlie, especially the latter, which has the less broken topography, the dust-storms are almost ceaseless, and bear forceful testimony to the amount of material which can be conveyed in the air. The wind careers over the country in gyrating whirls, to which the aborigines give the name of "willy-willy," nor have the white invaders ventured to call them otherwise; and as these whirlwinds go waltzing across the wretched town, they gather up in their skirts all the scattered refuse of a border civilization. I formed the impression that the wind had a prevailing direction from the southeast to the northwest, that is, from the nearest sea toward the heated interior; but the meteorological reports of the government do not confirm this supposition. In the accompanying table, Beaufort's scale of wind-force is employed. It will be observed that the meteorological reports confirm the impression that Kalgoorlie is more windy than Coolgardie, and that a quiet condition of the atmosphere is unusual in both districts. Nor is there any consistency of direction, as is proved by the observations made, for example, at

Kalgoorlie during the month of September, 1897. It is evident that the wind blew where it listed, and no man could tell whence it came. The following is Beaufort's scale of wind-force:

Number.	Description.	Speed of wind in miles per hour.
0	Calm.	3
1	Light air.	8
2	Light breeze.	13
3	Gentle breeze.	18
4	Moderate breeze.	23
5	Fresh breeze.	28
6	Strong breeze.	34
7	Moderate gale.	40
8	Fresh gale.	48
9	Strong gale.	56
10	Whole gale.	65
11	Storm.	75
12	Hurricane.	90

TABLE III.—*Wind-Force*, 1897.

Month.	Coolgardie.				Kalgoorlie.			
	9 A.M Max.	9 A.M Min.	3 P.M Max.	3 P.M Min.	9 A.M Max.	9 A.M Min.	3 P.M Max.	3 P.M Min.
January	3	1	3	0	2	1	3	3
February	4	2	4	1	7	2	7	2
March	3	2	3	1	5	2	2	0
April	4	0	3	2	2	1	9	1
May	8	2	3	0	5	2	2	2
June	2	1	3	2	9	2	5	2
July	5	1	6	1	3	2	4	2
August	5	1	6	1	6	2	8	2
September	9	1	9	1	6	2	9	2
October	5	1	8	1	7	2	6	2
November	6	1	9	1	5	2	8	2
December		1	4	1	2	2	3	2

TABLE IV.—*Variation in Direction of the Wind at Kalgoorlie during September*, 1897.

9 A.M. Direction of wind.	9 A.M. Days.	3 P.M. Direction of wind.	3 P.M. Days.
N.	4	N.	4
N. to E.	5	N. to E.	4
E.	3	E.	1
S. to E.	5	S. to E.	3
S.	1	S.	4
S. to W.	6	S. to W.	7
W.	1	W.	5
N. to W.	2	N. to W.	2

Observations such as have been quoted in Tables III. and IV. necessarily fail to record the really characteristic play of the wind in this region. The whirl-storms, referred to already, spring up suddenly, rush madly across the plain, suck up everything lying loose on the surface, and as suddenly subside. These apparently erratic air-disturbances are responsible for the transport of the greater part of the material which weathering and erosion have disintegrated. So far as I know, measurements of the transporting-power of the wind have not been made in this particular region; but elsewhere in Australia scattered observations have been made; for example, that fences 4 feet high are buried by drifting sand in a period only slightly exceeding two years. In the Libyan desert, bordering the Nile valley, the same results can be seen. Thus, for instance, the sand-storms bury the temple of the Sphinx every summer, and the road built by Ismail Pasha, from the Mena House to the Pyramids, is filled with sand up to the level of the 6-foot parapet in less than ten days.*

In West Australia there is much evidence of that which geologists euphuistically term the Æolian agency. The wind has been stirring and sifting the material lying loose on the surface until it has become classified to a remarkable degree. In traveling over the country, one is soon called upon to notice the broken white quartz scattered over the ground, in big patches many acres wide. These alternate with stretches, steel-gray in the morning and blue-black toward the close of day, of ironstone fragments. Leave the trail; go a short distance into the bush; and you will find the surface covered with dust in which each step leaves an evident footprint. It is the veritable dust of ages, not the earth-smoke blown from man's restless to-and-fro. The wind has sorted the quartz, the ironstone and the dust. The latter has been scattered in contemptuous carelessness all over the face of the weary desolation, but the heavy ironstone remains not far from where it was broken off the decomposing diorite until, shattered and comminuted to powder, it also is winnowed by the dust-storm. The numerous veins, large and small, traversing the country have contributed the quartz which the wind has collected, so that it sometimes covers the ground with the glittering whiteness of a snow-drift.

* So my dragoman informed me when I was there last February.

From these stretches of ironstone and quartz one would naturally infer the occurrence, somewhere underneath the surface, of large masses of both. Owing to the extreme slowness with which denudation progresses in this arid region, and the consequent very gradual lowering of the zone of oxidation, the rocks exhibit, above the drainage-level, a marked intensity of chemical action. The granite is kaolinized to an almost incoherent clay, and the diorite is rendered abnormally heavy in iron by the surface-concentration of decomposition-products. And, as the rock is eroded, the quartz, on account of its hardness, persists, so that a series of small stringers eventually yields an accumulation suggestive of its derivation from a large mass. It is a process of concentration which, there is reason to believe, has also affected the gold-occurrence, the upper portions of veins being enriched by the deposition of the gold left behind from lode-matter which was long ago disintegrated and removed by erosion.

If we now turn to the "dry-blower" and watch him at his work we shall see the same processes utilized in the winnowing of the gold.

III.—Dry-Blowing.

In West Australia the absence of running water renders unavailable the cradle and the sluice-box of ordinary placer-mining, with the result that the prospector has learnt, intuitively, to utilize the agency which he sees incessantly at work in the nature around him. Wind replaces water. The method is simple. Taking two pans,* he places one of them on the ground, empty, while into the other he puts a shovelful of the "dirt," that is, the sandy detritus containing the gold. The material is shaken up so as to bring the big lumps on top, and then, resting the pan on one knee and holding it with his left hand he uses the right hand to skim off the coarse particles (as shown in Fig. 5). Then standing erect and facing at right angles to the direction of the wind, he slowly empties the full pan into the empty one at his feet (see Fig. 6). As the stream of dry dirt falls, the wind selects the fine and blows it in a cloud of dust to leeward. The operation is then reversed, the pan which has just been emptied being placed on the ground so as

* The Australian calls them "dishes."

to receive the contents of the other. This is repeated three or four times, according to the degree of concentration effected. In a strong breeze one operation may prove sufficient. To prevent the loss of the fine gold which is sometimes carried away with the dust it is customary to spread a piece of canvas on the ground, one end being placed under the pan and the other extending to leeward.* The next stage is to further winnow the material by tossing it up and down in the pan (see Fig. 7); the latter is held slanting forward, and is jerked so as to throw the dirt from the front to the back of the pan. The light particles are separated, as chaff is driven from grain. Then, giving the dish a vanning movement, the prospector again removes the coarser particles that come to the surface by skimming them off with his hand. There now remains about half a pint of material, and this is diminished by panning, just as in water, the dry particles having a mobility permitting this method of treatment. Finally he drops on his knee, and, holding the pan (see Fig. 8) so that it is tilted forward, he raises it up to his mouth and uses the breath of his lungs to complete the process. The particles of gold are seen fringing the edge of the iron sand. If the yield consist of only a few minute particles,† he puts his moist thumb on them, and so transfers them to his pocket; but if there be any coarse pieces—nuggets—they are put into the leather wallet attached to his belt.

In watching a dry-blower at work, it becomes evident that the operation, like every process of concentration when properly conducted, consists of sizing and classification. The wind removes the fine sand and the dust, the operator's hand skims off the larger lumps of dirt, so that there finally remains a collection of those heavy particles of ironstone which, as in ordinary placer-mining, accompany the gold.

Owing to the perfect dryness of the dirt and the heat imparted to the surface of the iron pan under a tropical sun, the material behaves with much of the mobility which it would have if water and not air were the vehicle employed. It reminds one of the behavior of a charge in the roasting-furnace, in which the hot air cushions each particle so as to give to the mass an apparent fluidity.

* As illustrated in the Figs. 5, 6, 7, and 8. These were reproduced by Mr. Henry Reed, of Denver, from photographs taken by me at Kalgoorlie.

† "A few small colors," as we would put it in the United States.

The rapidity and completeness of the operation depend on the strength and uniformity of the wind. There is a constant light breeze on the gold-fields, even during those happy intervals when the dust-whirls have temporarily subsided; but the cloudy mornings of the wet season and the sultry days of the hottest summer months are alike unfavorable to dry-blowing, because at such times the air is dead. Many of the mining-camps are situated on a slight rise of ground, overlooking those desolate sinks of salt and sand which are called "lakes;" and the difference of level is marked by a constant breeze which is a good friend to the dry-blower.

In the history of ordinary alluvial mining, washing with the pan was succeeded by the use of the sluice-box and the cradle. Similarly the "dishes" of the dry-blower are replaced by machines of several types, all of which, however, are based on the idea of a shaking movement in the presence of a current of air. The simplest contrivance is represented in Fig. 9. This machine is 2 feet wide and 4 feet long. A is a hopper with a sheet-iron bottom punched with 1-inch holes, B is a 12-mesh screen, C is an 18-mesh screen, and D is the final product of the operation. The dry-blower empties a shovelful of dirt into the hopper, places his hands on the two sides of the machine and shakes it from side to side. There is sufficient play in the frame itself to permit a movement which causes the material to pass through the series of screens and accumulate underneath. It is then treated by hand, as previously described and illustrated. One man will put through about 5 tons of loose dirt in a working-day of seven hours.

Another and more elaborate contrivance is exhibited in Fig. 10. It consists of a series of four trays, hung on a triangular frame, B C D. The trays are 22 inches in diameter. They are comparatively flat and have screen-bottoms, through the center of which an iron rod passes to the eccentric, G, which receives the required movement through the lever, A E F, of which A is the handle. The trays are 5 inches apart and are held in place by wires, H H. The material to be treated is placed in the uppermost tray, which is a hopper pierced with inch-holes. No. 2 has $\frac{3}{8}$ holes, No. 3 has a 10-mesh screen and No. 4 has an 18-mesh screen. The lowest, No. 5, is flat and serves as a receptacle for the final product, which is dry-blown by hand, as heretofore described.

In these two contrivances no attempt is made to supplement the wind by an artificial air-current. The next step is to use a bellows. Fig. 11 shows such an arrangement.' It consists of a hopper, A, and a series of screens, B, E, F, G and H. By turning the handle, M, the disk, K, is revolved; and this, by means of a belt, transmits its movement to the pulley, which shakes the screen, B B, through the eccentric-rod, C, and at the same time operates the bellows, D, through the disk, K. Fig. 12 illustrates the machine when in operation.

When the material is placed in the hopper, A, and the machine is set in motion, the large lumps run off over the grizzly or sizing-screen, B B, the upper part of which is made of parallel wires $\frac{3}{16}$-inch apart, and the lower portion of 8-mesh. The finer stuff falls down into the shoots, C C and E E, respectively, and reaches F, which is another (12-mesh) screen, supplied also with riffles. As it descends through the screens at G and H, both 18-mesh, the blast from the bellows keeps the material in agitation, and aids the requisite separation between the particles of gold and the dust. The final product is panned by hand.

One of the most popular dry-blowing machines is that made by Steve Lorden, at Freemantle. It is illustrated in Fig. 13. The essential parts are:

A. Feed-hopper, sheet-iron bottom, punched with inch-holes, hinged at A_2 and provided with riffles, A_3, which arrest the heavier particles of gold, while the coarse lumps of dirt pass out of the machine over the shoot, B_2, and the fine stuff falls through into

B. Second hopper, which has riffles and smaller perforations, repeating the process. To examine this hopper, the upper one, A, is sprung out of position at A_4.

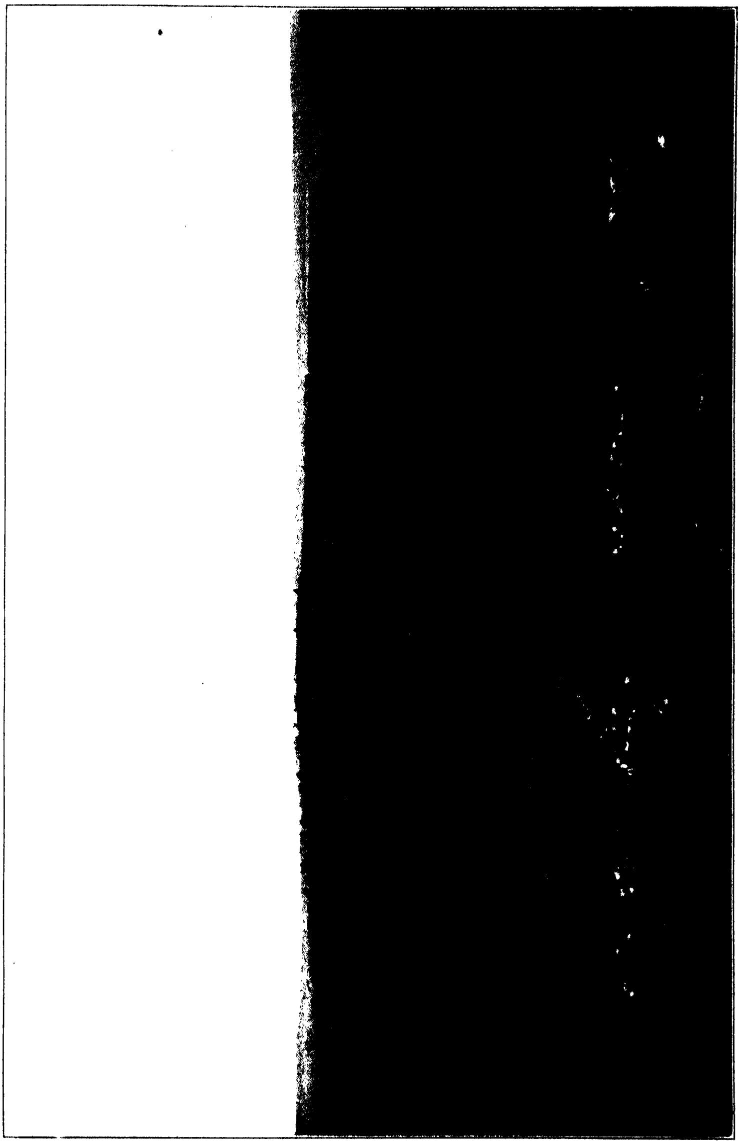
C, C. Return-shoots which lead the reduced material for further sizing upon the sloping screen, E, which also has a series of riffles, and is placed directly over the air-chamber, F.

D_2 is a discharge-shoot for the screen, E. G, G are air-channels from the bellows, H, H.

H, H. Double blast-bellows, one on each side, which ride on carriers, H_2, so arranged as to give the requisite play and to relieve the bellows from undue strain when in operation.

I, I. Rockers, bolted firmly to two foundation-blocks, I_2, which form the stand, the only part of the machine that is not

Fig. 2.

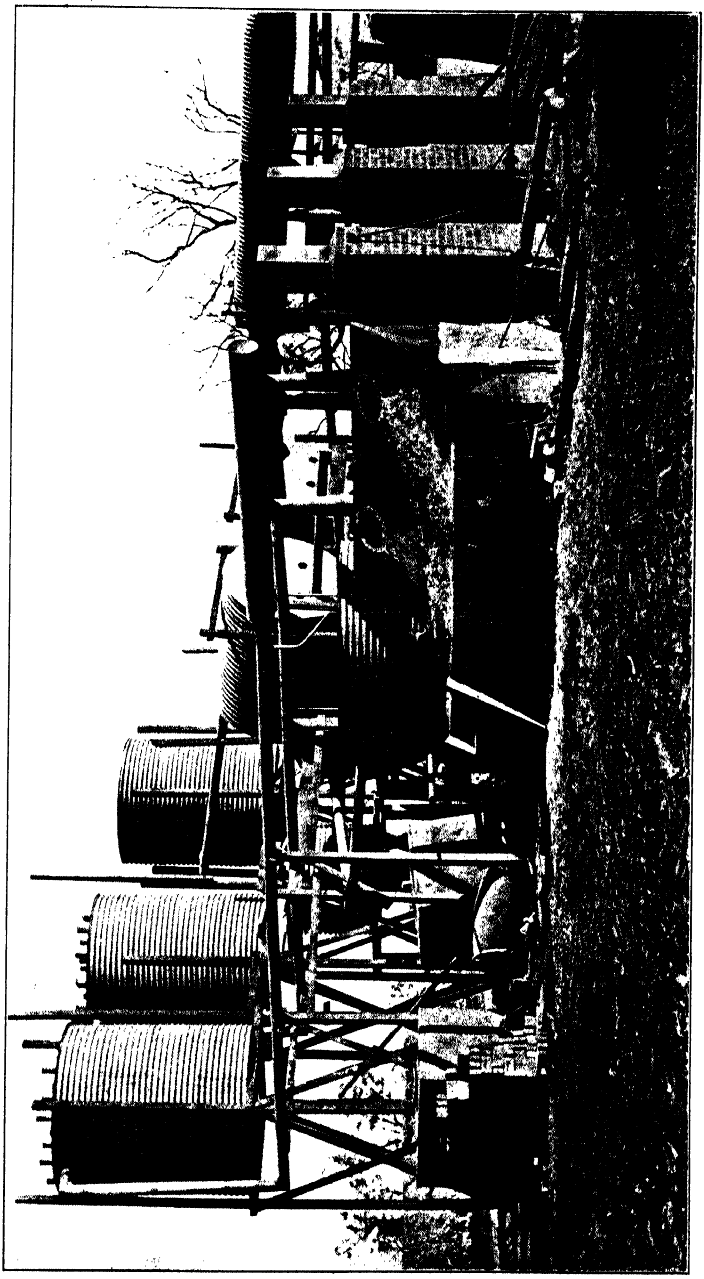


A Typical Scene in West Australia.

Fig. 3.

Buying Water at a Condenser.

FIG. 4.

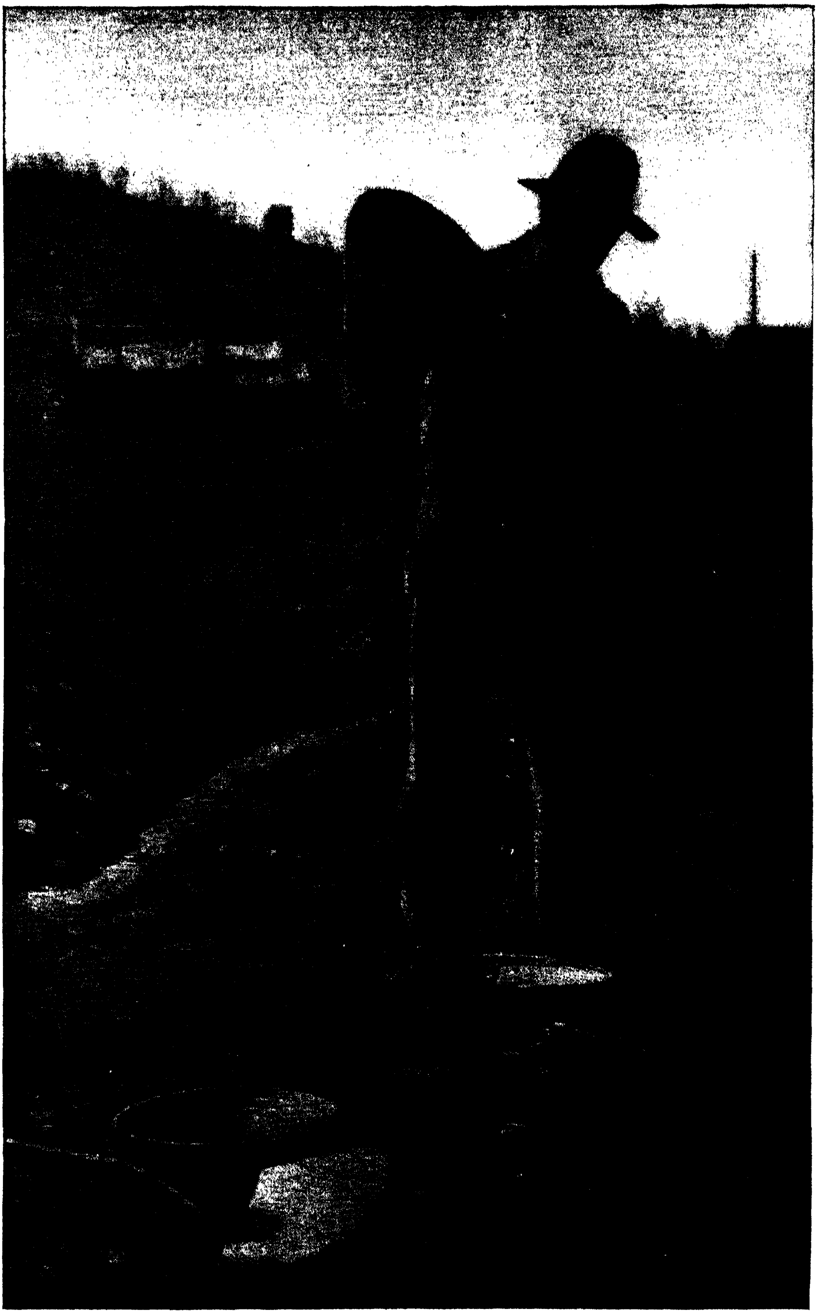


Condenser at the Lake View and Junction Mine, Kalgoorlie.

FIG. 5.

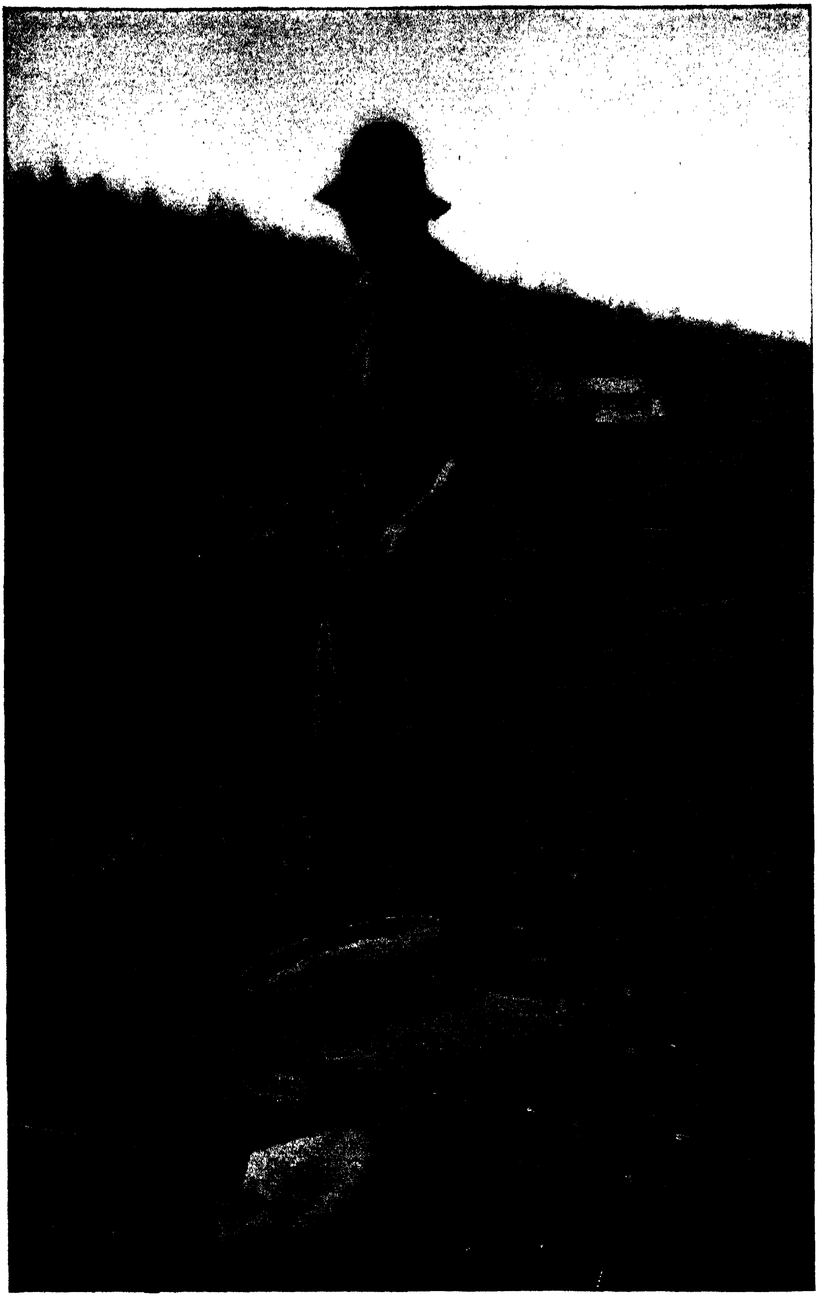
Dry-blower at Work.

FIG. 6.



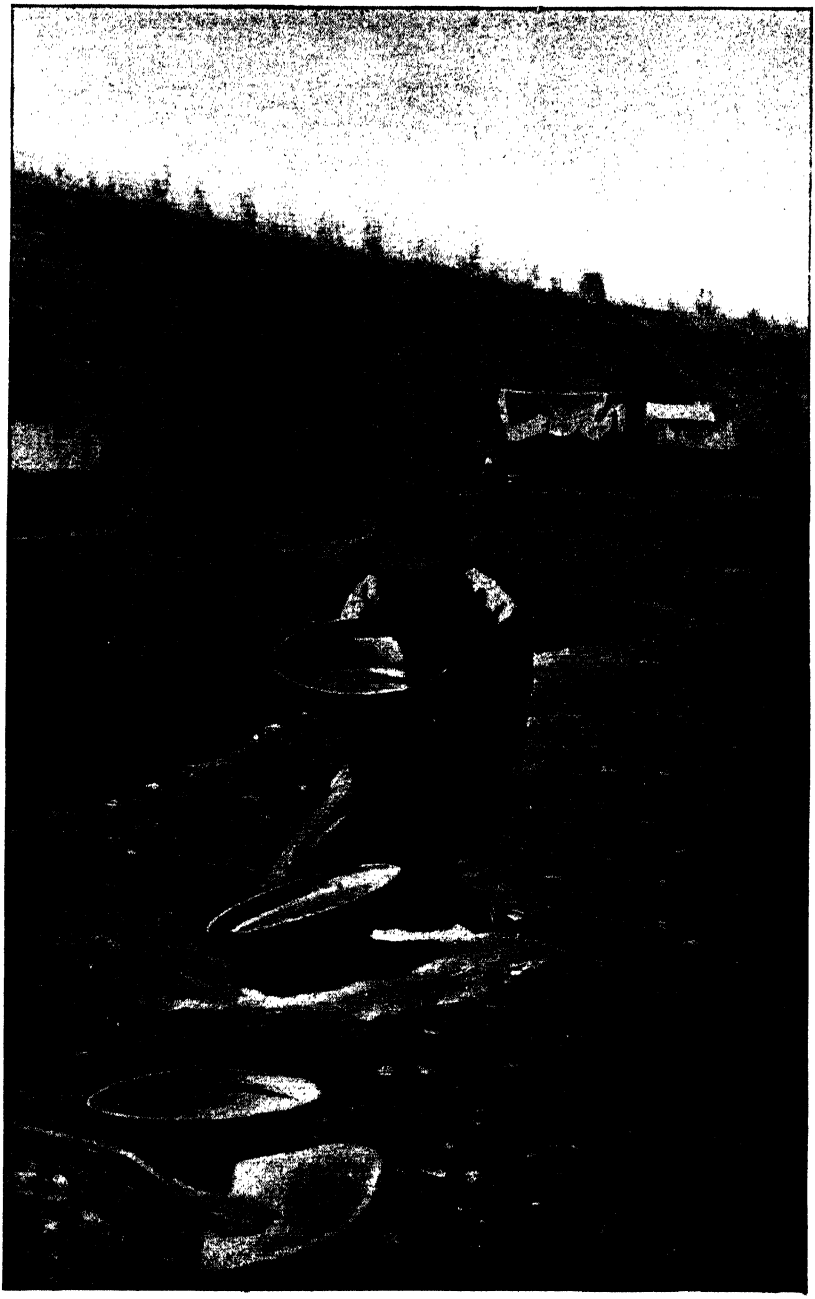
Dry-blower at Work.

FIG. 7.



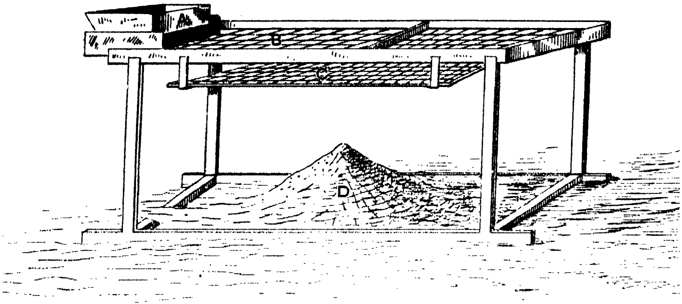
Dry-blower at Work.

FIG. 8.



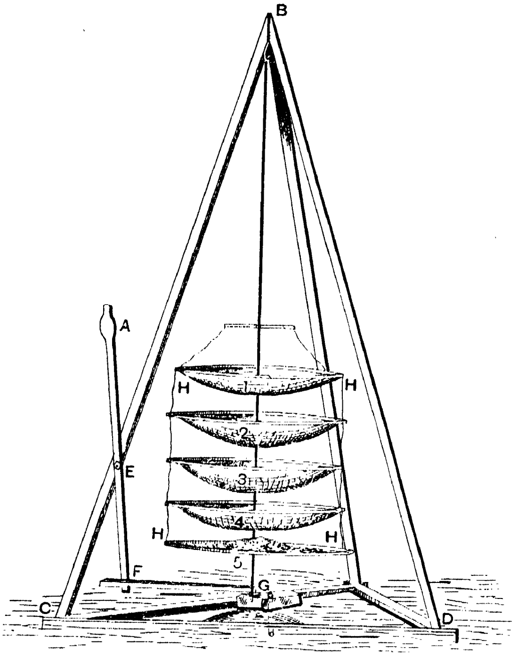
Dry-blower at Work.

Fig. 9.



DRY BLOWING MACHINE

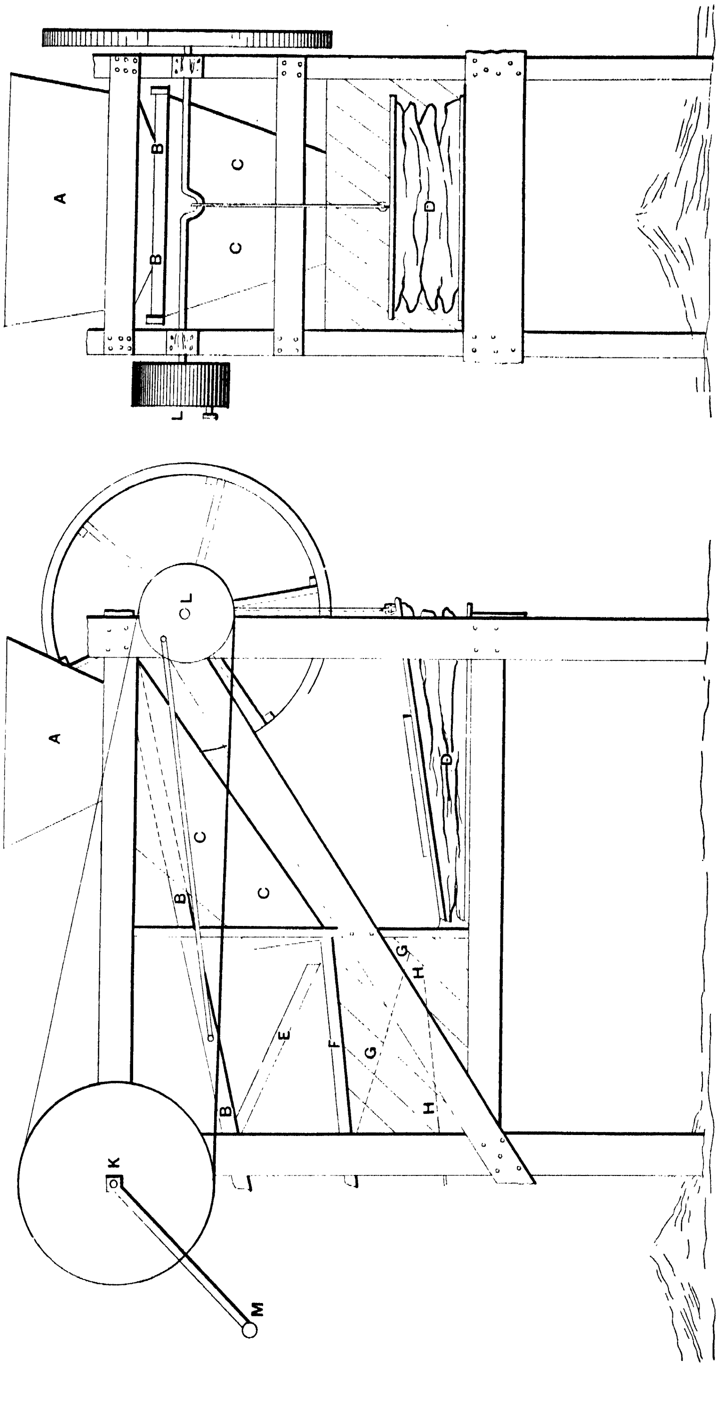
Fig. 10.



DRY BLOWING MACHINE

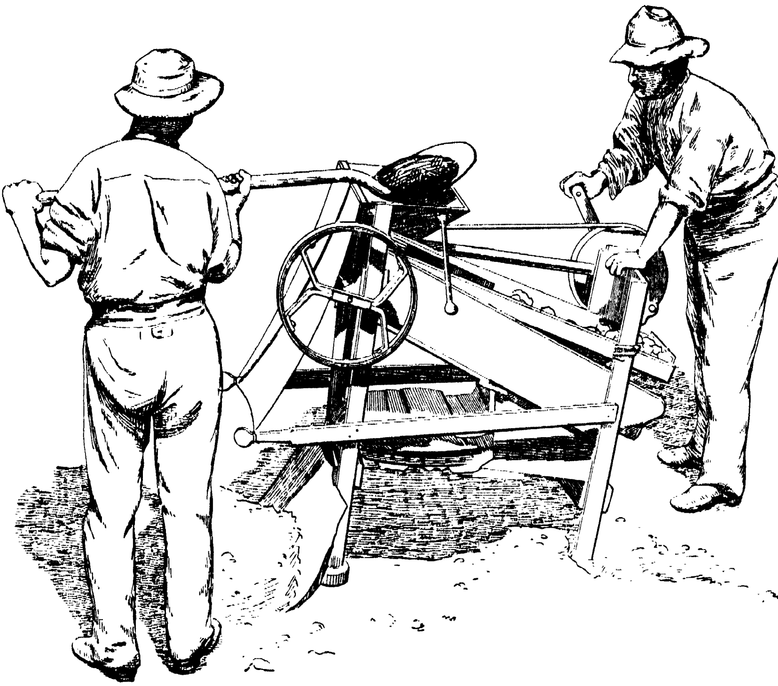
PLAN OF TRAY

Fig. 11.



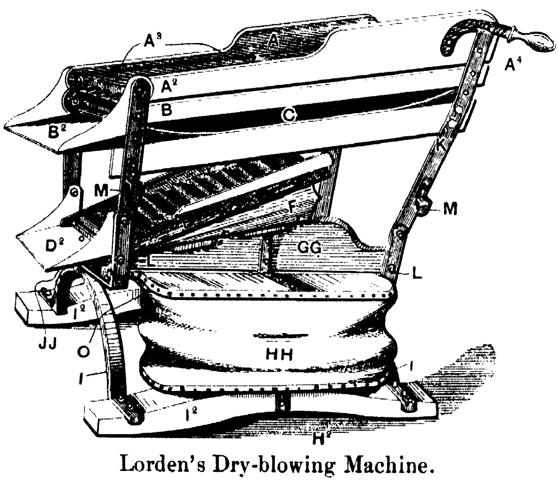
DRY BLOWING MACHINE
Scale 1" = 1 Foot

Fig. 12.



DRY BLOWERS AT WORK

Fig. 13.

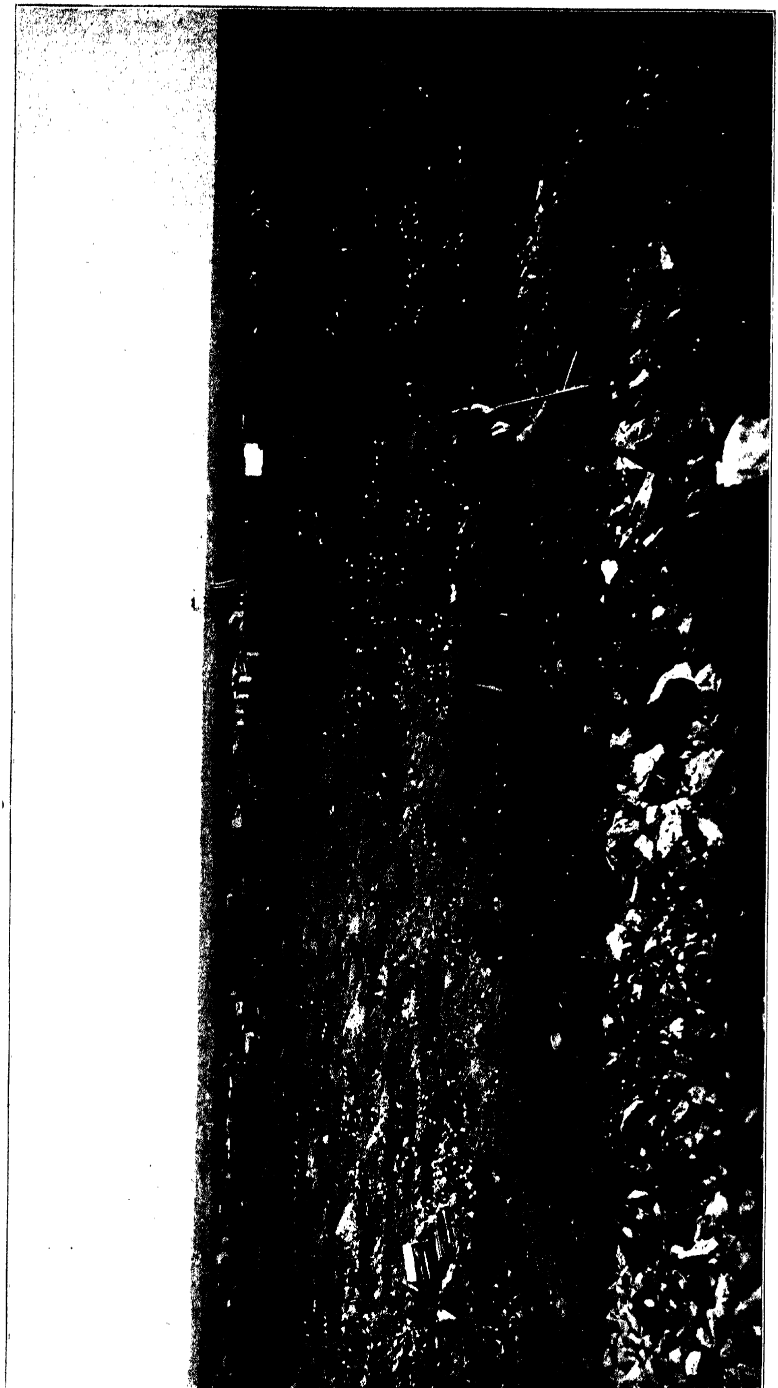


Lorden's Dry-blowing Machine.

Fig. 14.

Dry-blowers at Work.

FIG. 15.



Dry-blowers at Kalgoorlie, West Australia.

Fig. 16.

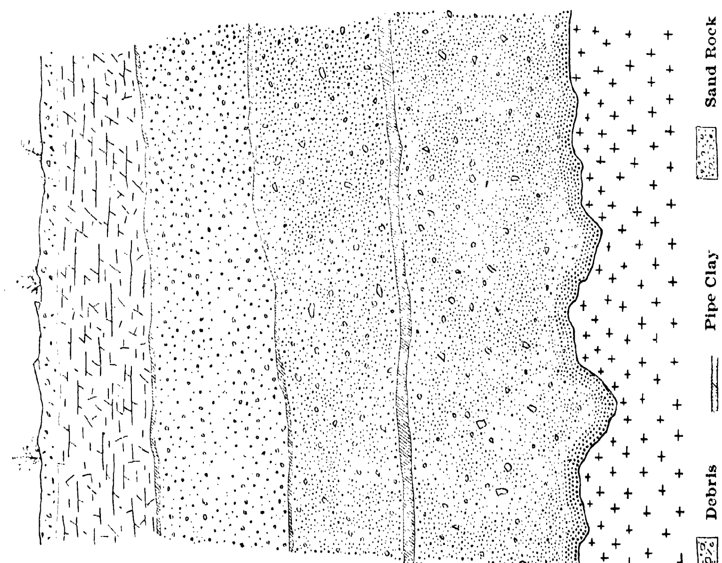
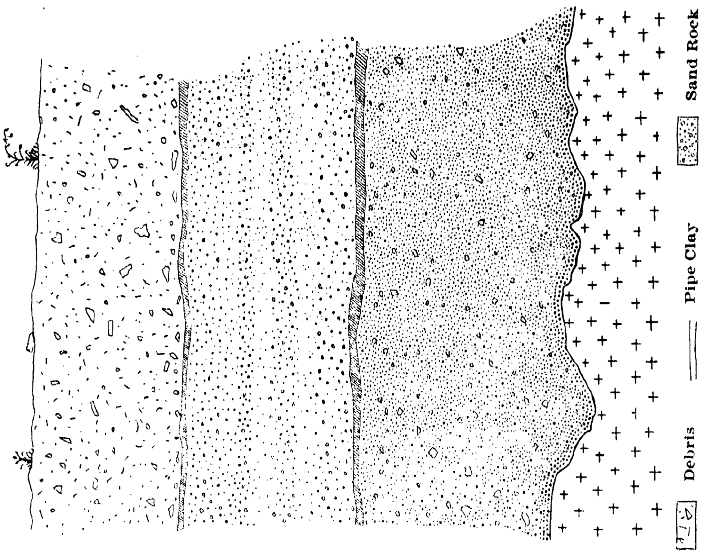
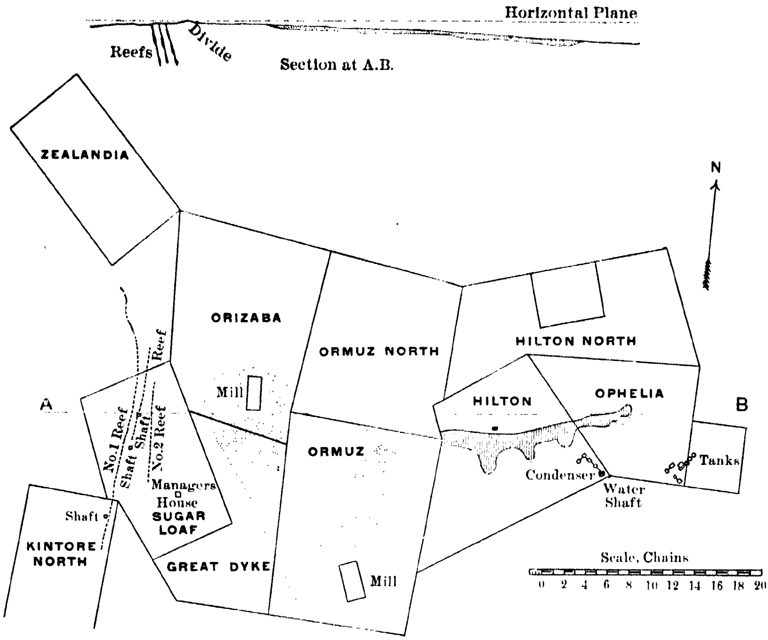


Fig. 17.



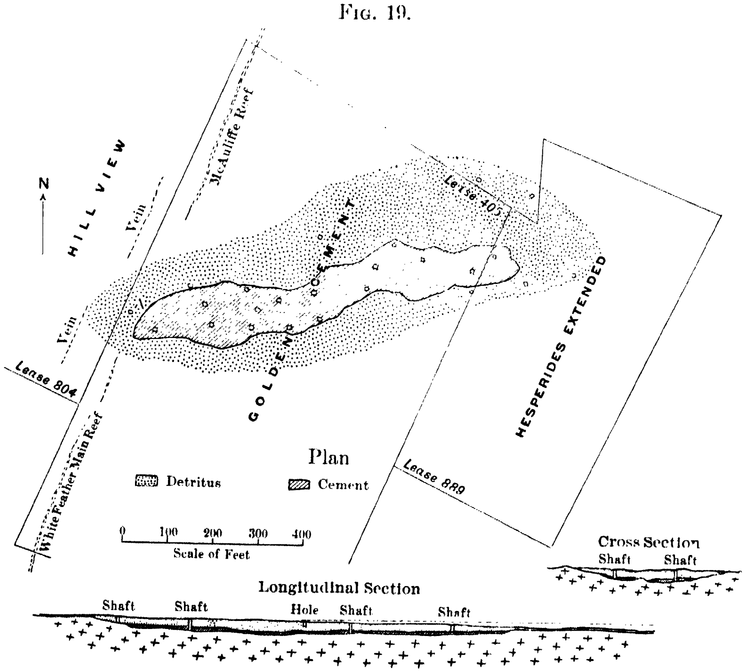
Sections in the Open Cuts at Kintore.

FIG. 18.



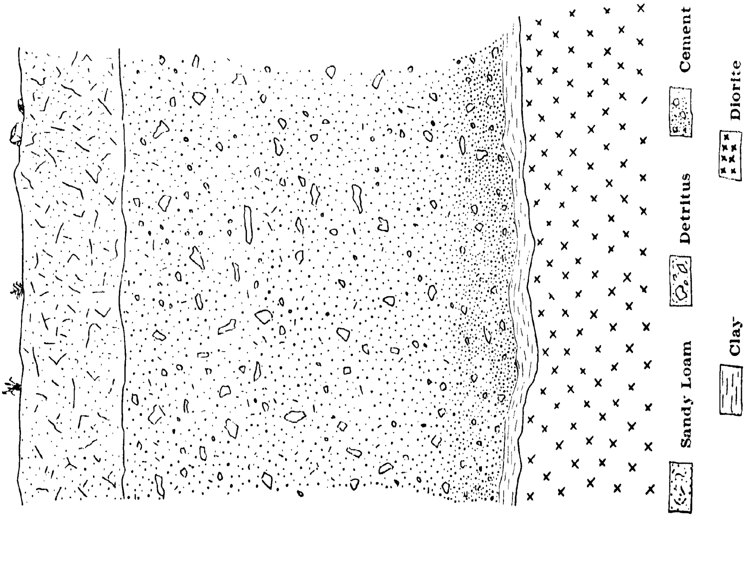
CEMENT DEPOSIT, KINTORE, WEST AUSTRALIA

FIG. 19.



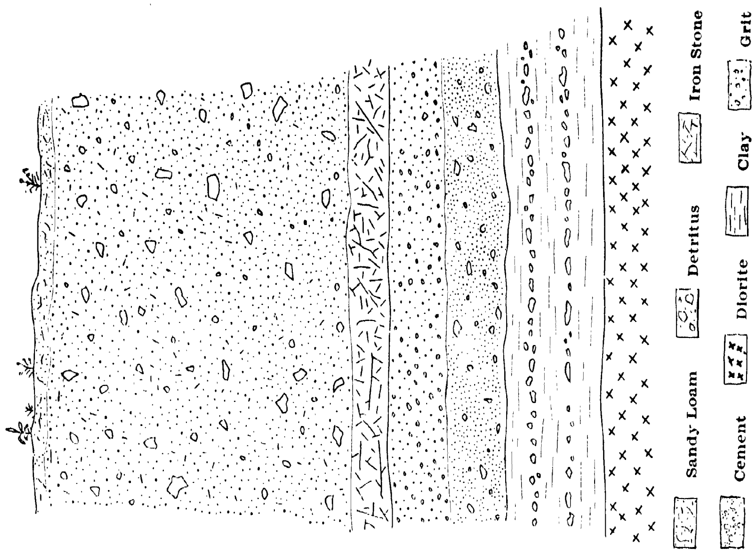
THE CEMENT DEPOSIT AT KANOWNA

FIG. 20.



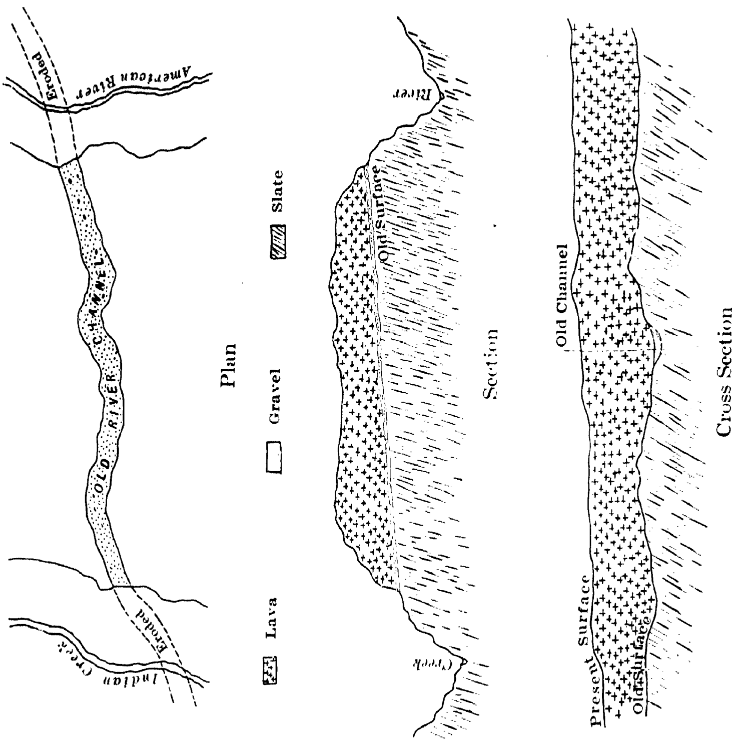
Typical Section at Kanowna.

FIG. 21.



Section in Fitzroy Cement.

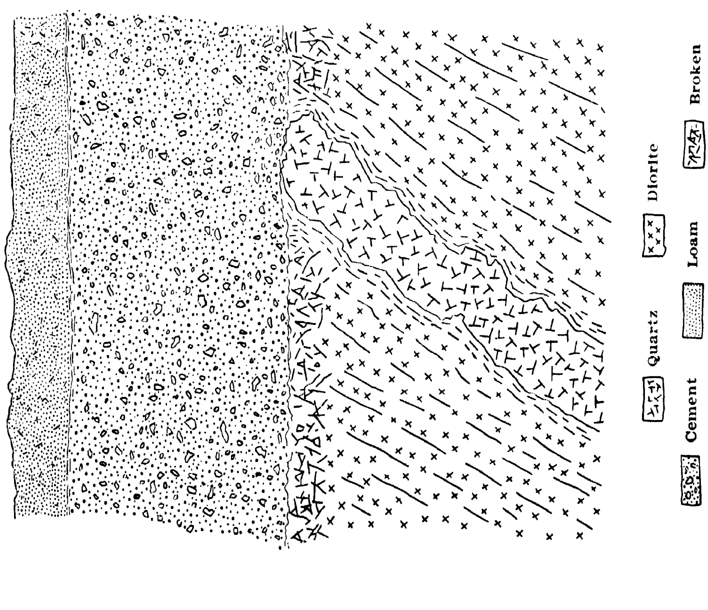
FIG. 23.



A TYPICAL DEEP LEAD

PLACER COUNTY, CALIFORNIA.

FIG. 22.



VEIN CAPPED BY ALLUVIUM

in motion. The curve of the rockers allows all dirt to fall away from the pivot-pins, J, J, by which the apparatus swings.

K, K. Standards, hinged at L, so that the machine folds up, as shown in Fig. 14.

M, M. Brackets for the insertion of two poles, by means of which a couple of men can carry the machine conveniently.

The operator holds one handle, at A_4, in each hand and rocks the machine, this serving simultaneously to put the bellows, hopper and screens all into movement. The machine weighs 124 pounds and has a capacity of from 10 to 14 loads (a load is roughly one ton) per day. The price at Freemantle is £16 or about $80.

The deposits exploited by the aid of these machines (see Fig. 15) are of a generally patchy character and lie at the upper ends of the depressions formed where the surface slopes away from ridges traversed by the veins of gold-bearing quartz. The prospector has an eye for the contour of the ground, and looks for the point where the rock-surface disappears under the fragmentary overburden which he calls "made ground" as distinguished from the underlying "bed-rock." In his search he is usually guided by seeing the outcrop of quartz, marking a possible source of detrital gold, and by the actual finding of specimens on the surface.

The distribution of the gold in these deposits reminds one of its position on a vanning-shovel. It may be traced up to the outcrop which yielded it, or it may be scattered in the sand for half a mile; but the rich and only workable part of the deposit will ordinarily be found at a distance of 30 or 40 feet from the reef.

Underneath these patches of superficial gold-bearing detritus there are found partially consolidated accumulations, which are more extensive and, quite apart from their greater economic value, are also of superior interest, because of their better-defined geological features.

IV.—The Cement-Deposits.

In the ordinary course of professional work I examined the two most important of these deposits, at Kintore and Kanowna, respectively. Since then a third formation of a similar character has been opened up, also at Kanowna. This I happened to see when in the stages of early development, and before it had been extensively exploited.

The deposit of cement at Kintore was one of the earliest worked. It is situated 23 miles northwest of Coolgardie, on the road to Menzies. The West Australian Proprietary Cement Company mined the ground with a success which was short-lived, because of the restricted quantity of material rich enough to pay the high costs of treatment. In four months, 7335 ounces of gold were obtained by treating 4115 tons in a stamp-mill, supplemented by cyanide-vats.

Enough work has been done to disclose the character of the deposit. Figs. 16 and 17 are representative sections obtained in the open cuts. Under a thin covering of sand and dust there occurs a bed of kaolin, ranging from a couple of inches to a foot in thickness; and this overlies from 15 inches to 2 feet of "sand-rock," which in turn gives place to the gold-bearing cement, which has an average thickness of 2½ feet. The last lies directly upon an irregular surface of decomposed granite.

The several layers composing the deposit are separated by seams of pipe-clay, which, like the kaolin, are simply the product of the decomposition of the constituents of the granite, particularly the feldspar. The sand-rock may be described as a coarse, incompletely consolidated sandstone or grit, consisting mainly of iron-stained particles of quartz, loosely cemented. The cement has a bluish-gray tinge, owing to the play of light on the quartz fragments. This, too, is not quite compacted, since fractures through the material do not break across the pebbles, which are harder than the clay binding them together. In this respect the cement differs, for example, from the South African "banket," to which it has been compared. From a distance, the cement looks like a coarse sandstone and exhibits a rough-joint structure. The materials of which it has been made up have undergone incipient sizing, so that different layers of varying coarseness are distinguishable.

The bed-rock is granite, so softened by decomposition as to be mistaken by the miners for a part of the alluvial deposit. It is kaolinized to a depth which the neighboring mine-shafts prove to reach a maximum of 130 feet. The surface on which the cement lies is marked by pot-holes having a maximum depth of 2 feet and a diameter reaching to 3 feet. These holes are filled with cement, which is usually poor save at the rims, where some of the richest mill-stuff has been obtained.

All the members of the deposit, from surface to bed-rock,

contain some gold, the kaolin being the poorest. The cement itself attains a maximum thickness of 5 feet. The richest parts occur in lateral embranchments from the main body of the deposit. The kaolin has become hardened and dried. It breaks like shale; and its true character is obscured by the down-filtering of red sand through cracks reaching to the soil overhead.

The deposit extends through a number of mining claims, as shown on the accompanying map (Fig. 18). It has been traced for a length of three-quarters of a mile. At the east end it begins as a narrow neck about 15 feet wide, and then enlarges steadily to 100 feet. Occasional bulges increase this width to a maximum of 250 feet. At the edge it gives place, as it thins out, to 2 or 3 feet of ironstone gravel, carrying 3 or 4 dwts. of gold per ton. The best part, economically, of the deposit lies in the Ophelia and Hilton claims, which, it will be noted, are situated at the lower end of the basin.

The bed-rock rises westward at the rate of 15 feet per thousand. This fact suggested that the origin of the gold-bearing cement was to be found in the reefs which were being profitably mined by the Sugar Loaf Company. The workings were 136 feet deep at the time of my visit in September, 1897. The veins traverse granite which has been kaolinized to 130 feet from the surface. They consist of white quartz and are narrow (4 to 12 inches), but they carry short shoots of very high-grade (3 to 10 ounces of gold per ton) ore. The gold occurs native, in flakes penetrating the cleavage-planes of the quartz like a golden mosaic, and also in coarser particles, which, under closer examination, prove to be octahedral crystals with curiously rounded edges. A comparison of these veins and their enclosing rock with the material composing the cement-deposit unquestionably indicates the derivation of the latter from the former. The cement carries gold which is exactly similar to that seen in the reefs; the quartz fragments in the alluvial are identical with the stone broken in the Sugar Loaf mine. Samples of both lie before me as I write, emphasizing the conclusion just stated. In the cement occur particles of quartz showing gold. The loose gold in the cement has been but slightly worn, and the quartz pieces are rather subangular than rounded, so that they can hardly be termed "pebbles;" and the deposit itself is better defined as an agglomerate than as a conglomerate.

The binding-material, the overlying layer of kaolin and the sand-rock capping the gold-bearing stratum of cement, all exhibit very clearly their derivation from a decomposed granite, similar to that which encloses the reefs and forms the bed-rock of the alluvium itself.

The topography confirms this supposed relationship. The highest point along the major axis of the cement-deposit is a very low ridge separating the workings of the Sugar Loaf from the alluvial ground. The house of the manager of the Sugar Loaf is on this divide. The reefs are 462 feet westward, and only 15 feet lower where they appear at the surface. The cement deposit begins on the Great Dyke lease, at a point 530 feet eastward, and only 8 feet lower. The cement then extends on a gentle down-slope of 15 feet per thousand for a distance of 3500 feet.*

It occupies a very shallow depression, and in its structure bears internal evidence of considerable geological age, suggesting that it was formed at a time when the Sugar Loaf reefs reached the surface at a level superior to the slight ridge now separating them. Reference to the map and longitudinal section (Fig. 18) will aid the above description.

Another deposit of similar character has been explored at Kanowna, 25 miles northeast of Kalgoorlie, and about 60 miles east of the locality just described.

The discovery was made in 1893. Each digger secured a claim 50 feet square, and sunk a shaft to the gold-bearing cement which the dry-blowers had uncovered in the course of their prospecting. The deposit became in due time "gophered" with holes and shafts, so that the boundaries of the cement were accurately determined. In 1895 an English company secured the property and consolidated the claims into larger leases. The expectations held out freely by responsible engineers that an extremely profitable enterprise could be based on the remnants of gold-bearing ground were wholly dissipated in the succeeding two years.†

The deposit lies in a shallow trough, the longer axis lying

* I am indebted to the courtesy of Mr. Alexander Brand and Mr. T. G. Paisley, the managers of the two companies, for the measurements quoted.

† A gross blunder was made, simply through insufficient and unsystematic sampling.

east and west. The body of gold-bearing cement has a length of about 700 feet and a maximum width of 105 feet (see Fig. 19). Vertical sections exhibit an overburden of sandy loam, from a few inches to 2½ feet thick. This was the material worked by the dry-blowers. Then comes a layer of detritus, called "wash" by the miners, composed of fragments of ironstone and quartz imbedded in clay, and reaching to a maximum of 25 feet from the surface. This overlies the cement itself, 6 inches to 5 feet thick, and easily distinguished from its roof of detritus and its floor of clay. The cement consists of particles of quartz in a greenish clay. Near the rim of the trough the quartz occurs in larger and more angular pieces. A typical section, obtained from a pillar in the old workings, is given in Fig. 20.

The gold-contents are irregular. The whole body of cement probably averaged one ounce per ton; but only the richest parts were worked, and these carried many ounces to the ton; so that the remnants now accessible average, from the surface down, about 3½ dwts.* The clay carries 2 dwts. per ton. The material was treated at neighboring stamp-mills.

When the neighboring mines, the White Feather Reward and the White Feather Main Reef, were visited, it seemed as natural to deduce this cement-deposit from the erosion of gold-bearing quartz-veins as it had been to relate the Sugar Loaf reefs at Kintore with the deposit worked by the West Australian Proprietary Cement Company. Further investigation confirmed this inference.

The McAuliffe vein (worked by the W. F. Reward mine) and the Main Reef (worked by the W. F. Main Reef mine) traverse diorite at or near the line where large dikes of granite-porphyry penetrate. The two reefs are probably identical, and have a strike which takes them right across the longer axis of the cement-deposit at a point near the head of the trough in which it lies (see map, Fig. 19). A shaft recently sunk to a depth of 200 feet by the Golden Cement Company, at a point marked A, reached this reef by means of a crosscut, and found a comparatively barren quartz-vein, carrying small spots of rich ore. The enclosing rock was diorite, and the quartz itself was encased on both sides by bands of clay fully 2 feet thick.

* Information which I owe to the courtesy of Mr. S. H. Williams, the manager

On comparing the veins and their encasing rock, as seen in the workings of the two mines on opposite sides of the alluvial deposit, it is not found necessary to go further for the origin of the latter. The cement is underlain by a clay which is essentially steatite, and is as readily traceable to the neighboring diorite as the kaolin at Kintore was deducible from the granite. The green color of the cement is imparted by chlorite, derived from the decomposition of the epidote in the diorite. The "ironstone" of the detritus overlying the gold-bearing part of the deposit consists of fragments of altered diorite. The quartz in the cement and the gold accompanying it are both identical with those of the reefs close by.

Here also the topography confirms the suggested explanation. The cement lies in a shallow depression, at the upper rim of which the quartz reefs cross the country. Furthermore, these reefs traverse a low divide, which in a rough way separates the deposit from another, which slopes in the opposite direction. The latter is known as the Fitzroy cement. In this deposit rich discoveries were made during October, 1897. A typical section is exhibited in Fig. 21.

Apart from their importance as depositories of gold, the cements have played an interesting part in the development of the gold-fields, because they often cover the tops of reefs. In Fig. 22 such an occurrence* is illustrated. A similar feature proved a serious hindrance to the recognition of the lodes at Kalgoorlie, where worthless quartz veins were worked for some time before a trench traversing the cap of cement accidentally uncovered the top of one of the rich deposits of telluride ore, which did not outcrop, on account of the comparative softness of the lode.

V.—Theories of Origin.

Of course, several theories have been mooted, the most fanciful of which have naturally been those of the working miner himself. The fact that the gold-bearing cement is in places overlain by a considerable thickness of partially-consolidated rock has led to the supposition that the deposit was a "deep lead;" while the resemblance to a conglomerate has caused more than one Africander to liken it to the "banket" of the Transvaal.

* It is the top of the Lady Shenton reef, at Menzies.

The latter is an immense shore-deposit of gold-bearing conglomerates, now covered by a series of later sediments. Its dimensions, comparative homogeneity and persistence are in striking contrast to the narrow, restricted, irregular patches of detritus which have been described as occurring in West Australia. This total unlikeness renders unnecessary any discussion of a fancied similarity of origin.

But because this alluvium disappears under an overburden of rock,* the Australian digger easily fancies he is working a deposit similar to the "deep leads" with which he became familiar at Ballarat, for example. A distinguished government geologist from a neighboring colony visited Kanowna in October, 1897, and gave authority to the term "deep lead" by using it himself. "Deep" it may be, for that is a comparative adjective, but a "deep lead" in the technical sense it most assuredly is not. On the Forest Hill divide,† in Placer county, California, and at Creswick, in the Ballarat district, Victoria, the typical "deep leads" occur. They are, as is well known, old (Miocene) gold-bearing river channels, which have been saved from erosion by a cap of lava. The lava probably overflowed the original surface as a steaming mud, and is now found consolidated into a volcanic rock sufficiently hard to need little timbering when penetrated by underground workings. The cement deposits of West Australia occur under an overburden of "made ground;" that is to say, both the deposit itself and all the material under which it dips are of distinctly detrital origin, the products of weathering and erosion accumulated in shallow depressions of the much-decomposed surface of granite or diorite.

It is the placer of a country destitute of running water. The climatic conditions and the physiography of the Coolgardie gold-fields have been carefully described, in order to make it evident why these deposits differ from those of more favored countries, like California or New Zealand. Surely it is not in keeping with the scientific method to seek for fantastic or far-fetched explanations, when processes in operation to-day

* Using the word in its geological sense. Mud is "rock" as much as granite.

† I append a drawing (Fig. 23) of a typical deep-lead recently examined by me in this particular locality.

are able to supply an adequate understanding of the observed facts.

The quartz of the cement is subangular; it has evidently undergone very little attrition, and suggests, therefore, that it has not traveled far. On comparing it with the matrix of neighboring veins, an identity appears obvious. The examination of the topography renders highly probable the derivation of the one from the other. The cementing material is similarly found to be the clay resulting from the decomposition of the rock encasing the quartz-veins, and varying according to the composition of that rock, whether it be granite or diorite. Finally, the gold particles which have rendered the cement worth mining are found to be identical in fineness and physical appearance with the gold of the neighboring veins, and their scarcely-rounded edges invite the conclusion that the gold also has not been borne far from the place of its origin. The comparatively unclassified condition of the deposits is in keeping with the evidence afforded by the material of which they are composed. The absence of running water on this desert plateau has prevented any such sifting-process as in other regions leads to the deposition of well-defined layers of clay, gravel and gold upon a clean bed-rock. It is an exceptional illustration of the working of those agencies to whose unceasing play is due the configuration of the earth's surface; it is geological action in its most instructive form.

VI.—Water-Supply.

The early history of the gold-fields of West Australia is the record of a struggle to exist amid conditions which were inimical to human life on account of the scarcity of water. That great necessity has been, in some sort, satisfied by the energetic action of the government, supplemented by private enterprise. The gold-fields are now dotted over with condensing-plants, which turn the brine of the wells into water fitted for the use of man and beast. Existence is thus rendered endurable; but the mining industry is still handicapped by an item of cost unknown in more favored regions.

The water of the country is salt, sometimes almost to the point of saturation. Sea-water contains 3½ per cent. of salts, three-quarters of this percentage being common salt, the

chloride of sodium. At Menzies, in September, 1897, I found one* of the two important mines of that district using water which contained 17 per cent. of salts, and the manager informed me that in December evaporation increased the amount to 30 per cent.† For this liquid he paid 25 shillings per thousand gallons. It came from a neighboring "soak." The condensed (distilled) water, bought for use in the boilers, cost £1 per hundred gallons. Milling in a ten-stamp-mill was carried on at an average cost of 30 shillings or $6 per ton, the item of water alone amounting to 13 shillings or $3.25 per ton.

Under these conditions a wet mine becomes a source of revenue. Many properties at Kalgoorlie, unable to find pay-ore, lessened the expenses of development by selling their water to those that had mills. The price varied according to the season. At the Great Boulder Main Reef mine, for example, the lowest price paid for water during 1897 was £3.10s. per thousand gallons, and the highest £11.5s. This was piped from neighboring shafts, and had not passed through the condenser; it therefore had the character of sea-water, but it was four times as saline.

An analysis of the water of the Great Boulder Proprietary mine gave the following results. The sample was turbid, and it was found that the matter in suspension amounted to 5.25 grains per gallon, or .075 gramme per liter. The clear water on analysis yielded:

	Grammes per liter.
Silica (SiO_2),	0.038
Alumina and ferric oxide (Al_2O_3 and Fe_2O_3),	0.024
Lime (CaO),	1.878
Magnesia (MgO),	8.106
Soda (Na_2O),	48.470
Carbonic anhydride (CO_2),	0.064
Sulphuric anhydride (SO_3),	6.026
Chlorine (Cl),	67.230
	131.836
Deduct oxygen equivalent to chlorine,	15.150
	116.686
Combined water, organic matter and loss,	8.534
Total solids,	125.220

* The Queensland Menzies mine.

† The water of the Dead Sea varies from 20 to 26 per cent. salts, and of this, 10 per cent is common salt.

The chief salts probably present were, therefore:

	Grammes per liter.
Calcium carbonate, $CaCO_3$,	0.145
Calcium sulphate, $CaSO_4$,	4.365
Magnesium sulphate, $MgSO_4$,	5.189
Magnesium chloride, $MgCl_2$,	15.144
Sodium chloride, NaCl,	91.467

Expressed in grains per gallon, the results appear more striking. The proportion of common salt amounts to no less than 6402.7 grains per gallon. Ordinary drinking-water contains about 3 grains of common salt per gallon.

The water-supply of the region comes from two sources, namely, that which has collected amid the purely superficial deposits of *débris* and drift covering the actual rock-surface, and, secondly, that which has penetrated through the decomposed rock down to the zone where oxidation ceases.

Wherever a depression occurs, the prevailing rocks, granite and diorite, are overlaid with a variable thickness of their own detritus, which allows the collection of rain-water and affords protection from too rapid evaporation. These are known as "soaks." The government geologist defines them as "valleys silted up with a thin covering of recent superficial deposits more or less saturated with water."* At Hampton Plains, 8 miles from Coolgardie, a supply of condensing-water has been obtained from beds of this nature. The section† was as follows:

	Thickness. Feet.	Depth. Feet.
Clay, with ironstone gravel,	27	27
Fine sand,	30	57
Coarse yellow sand,	43	100
Clay,	4	104
Land wash,	11	115
Kaolin,	8	123
Bed-rock,	39	162

Water was struck in the third stratum, described as coarse yellow sand. The bed-rock was granite.

During the wet season, some of the depressions filled with such accumulations of detritus will receive more water than

* Report in connection with the water-supply of the gold-fields. 1897. A. Gibb Maitland.

† From the report just referred to.

they can hold, and then the eye becomes gladdened for a few days by the sight of water running over the surface. As it becomes diminished by evaporation it disappears from view, but will be found to linger in the rock-holes, where the supply is maintained by the slow drainage of the surrounding porous area. These are called by the aborigines "guamma" holes. The life of such natural wells depends, of course, on the dimensions of the water-bearing depression and upon the relative porosity of the deposit which it drains.

These supplies are in their nature uncertain. The next and more important source of supply is found at the ordinary drainage-level of the country, namely, the horizon where the oxidation of the surface ceases and the relatively hard unoxidized rock offers a partial barrier to the free descent of the waters which have percolated through the overlying formation. The depth of this zone will depend upon the permeability of the superficial rock-formation; it varies from 40 feet at Earlston to 202 feet at Kalgoorlie. A characteristic section is that given by the well put down on Reserve 3096, Coolgardie, where the Gold-fields Water Supply Department sunk 165 feet and found 7 feet of sand, 47 feet of conglomerate and 111 feet of decomposed granite. The water was found at the base of the last, just above the unaltered granite.

Condensers.—Frequent mention has been made of the condenser. This is a characteristic feature of every mining settlement in the interior of Western Australia, and occupies the position accorded to the public well of a European village. Without this process of distillation, which renders the brackish water of the wilderness fit for human consumption, the mining industry could never have progressed beyond the merest prospecting.

The introduction of condensers is traceable to the sailors who took a prominent part in the early exploring expeditions. The name itself suggests this; for a landsman would be likely to call the condenser a "still." When the rush to the new gold-fields occurred, the government, by erecting condensers at intervals along the main lines of travel, did much to diminish the loss of life otherwise inevitable in times of wild excitement by reason of the recklessness of those who joined the stampede without due care for the great necessity of life in a tropical desert.

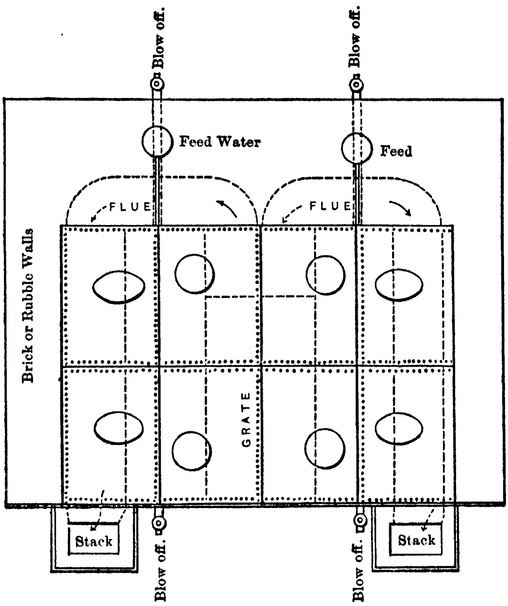
The process of converting brine into drinkable water is simple. The salt water is put into a boiler and converted into steam, which is then condensed in vessels presenting a maximum of cooling-surface. Ship-tanks, having a capacity of 400 gallons, are commonly employed as boilers, and the condensing apparatus is constructed out of the corrugated iron which is everywhere employed for roofing-purposes. The tanks used as boilers are usually set on edge in pairs, as shown in Figs. 24 and 25. The average product of each 400-gallon tank is about 300 gallons of distilled water daily. Two vertical short iron pipes draw off the steam, which passes into condensing chambers or "coolers." The latter were originally plain circular tanks, which were increased in capacity by the addition of sections, increasing the height. Sheets of corrugated iron were bent round until the ends met, and these were united by riveting and soldering. Several such sections were united, and a tubular tower, about 30 feet high and 3 feet in diameter, was the result. The top and bottom were closed by a flat sheet of iron. A 6-inch pipe connected the towers, and steam traveled up the one tower and down the next.

This type has been superseded of late by annular chambers. An outer corrugated iron cylinder, 2.5 feet in diameter, surrounds an inner 1.5 ft. cylinder, so as to leave an annular space, 6 inches wide, which becomes the condensing-chamber. No attempt is made to supplement the cooling effect of the surrounding air, though a brush shelter is sometimes erected so as to ward off the direct rays of the sun.*

The daily expenditure includes the labor of two men, one on each shift, and the cost of the fuel consumed. A typical condenser is that erected on the Lake View and Boulder Junction mine, shown in Fig. 4. This plant has a capacity of 1,500 gallons of condensed water per day and cost £100. The water treated comes from the mine and shaft and has a specific gravity of 1.03385, the total solids amounting, according to the analyses of Mr. E. S. Simpson, to 4.9308 per cent. and the chlorine to 2.3933 per cent. The cost averages, during the cool season, 5 shillings, and during the summer 6.5 shillings

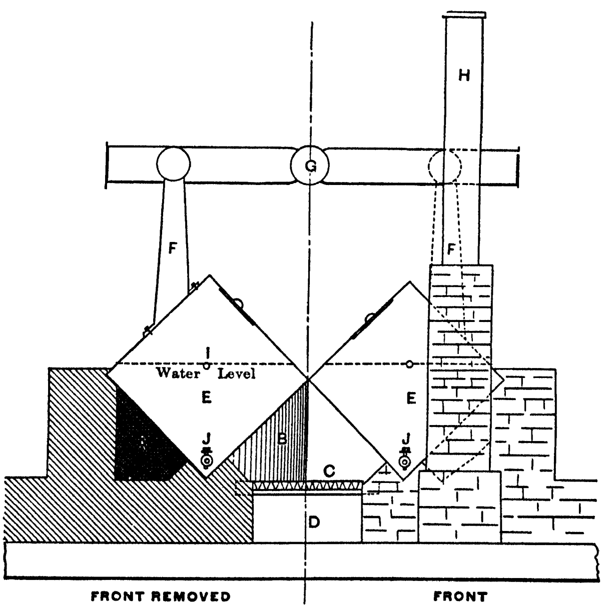
* For many of these details I am indebted to Mr. Frank G. Grace, Kalgoorlie, and to Mr. Edward S. Simpson, Government Assayer, Perth.

Fig. 24.



Plan of Ordinary Condenser.

Fig. 25.



Section and Elevation of Ordinary Condenser.

per hundred gallons of condensed water. The condensed water is sold at from 10 to 12 shillings per hundred gallons. A first-class condenser would consist of at least 8 boilers, each having 400 gallons capacity and a daily output of 300 gallons of condensed water, giving the plant a daily yield of 2,400 gallons. The salt water is usually bought for from 2 to 4 shillings per thousand gallons. Fuel costs 20 shillings per cord, and is consumed at the rate of 1 cord per thousand gallons of condensed water. Thus the cost would be:

2 men at £4 per week of 6 days, ...	27 shillings.
2½ cords of wood, ...	50 "
3,200 gallons of salt water, ...	10 "
Total, ...	87 shillings.

This would be at the rate of 36 shillings per thousand gallons. During November, 1897, condensed water sold for 100 to 150 shillings per thousand gallons.

Concerning the use of salt water in the stamp-mills and leaching works of Western Australia, I would say that its density is an obstacle to amalgamation because of the facility with which slimes are created. The finer particles of gold are thus prevented from settling on the copper plates of the tables, and are carried away in the tailings. At Kalgoorlie the cyanide works use the natural brine successfully; its magnesia being precipitated by lime, so as to prevent the decomposition of the stock-solution.

In order to aid the development of the mining industry, which is still severely handicapped by the want of a sufficient supply of water, the government of West Australia has decided to carry out a hydraulic enterprise of great magnitude. It is proposed to supply 5,000,000 gallons of fresh water per day to the Coolgardie gold-field by building a pipe-line from the Darling range, where the Helena river will be impounded by a concrete dam 100 feet in height and 650 feet long. The source of supply is 320 feet above sea-level, while the service-reservoir at Coolgardie will be on Mt. Burgess, at a height of 1670 feet, or 1350 feet higher. These two reservoirs will be connected by 330 miles of 30-inch steel-pipe. Nine pumping-stations will be required. The appropriation for this work is $12,500,000. The annual operating expenses will probably

approximate $1,600,000, and water will be delivered through a hundred miles of distributing pipes at a cost of 3 shillings and 6 pence, or 84 cents, per thousand gallons.

It will occur to the reader to ask whether boring for artesian wells has been attempted. Yes; in obedience to the public demand, the government put down a bore at Coolgardie which reached a depth exceeding 2000 feet and found—granite. The geological conditions render an artesian flow of water highly improbable. Nevertheless, in this colony, as elsewhere among the arid tracts of Australia, there is a whispered hope of finding a subterranean river. As the Carson and the Humboldt are swallowed up by the alkali wastes of Nevada, so in the desert plains of this southern continent there are many streams which flow into the interior and lose themselves in the sand or find for themselves an undergound channel.* This fact has given rise to conceptions, more poetic than scientific, of a great subterranean river yet to be discovered, and destined in days to come to make the desert break forth into fertility. It is a dream. No irrigation can turn the wastes of quartz-sand into waving fields of wheat. Time, geological time, covering a period to measure which the duration of a man's life is an inadequate unit, can alone render the wilderness fit for human habitation.

Mineral Lode-Locations in British Columbia.

BY WILLIAM BRADEN, HELENA, MONTANA.

(Buffalo Meeting, October, 1898.)

In view of the current discussion of a proposed change in the United States mining law, abolishing the feature known as the extralateral right of a lode-location, it is an interesting circumstance that in the neighboring Province of British Columbia this feature was tried for eight years and then abandoned. The results of that abandonment have been such as to disprove the proposition, so confidently advanced by many

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* This occurs in Queensland, where the geological conditions are quite different. Several very successful bore-holes have been put down, an abundant artesian flow being obtained at depths of 2000 feet and over.

writers, that there is something in the nature of the mineral deposits of the Pacific Coast which requires the grant of the extralateral right to mining locators. In abolishing this peculiar privilege, British Columbia has followed the example of all civilized countries except our own.

From 1884 to 1892 the Mineral Act of this Province was modeled after that of the United States, and authorized lode-locations 1500 feet long by 600 feet wide, carrying the extra-lateral right. But it was soon realized that, however equitable in intention, this principle was so liable to complications in application and such an endless source of litigation as to be a greater injury than aid to mining. The Act was consequently revised in 1891, and, since the end of that year, the Provincial lode-law (further amended in 1896 and 1897) has been liberal to the locator and miner, and clear and simple of execution.

As it stands to-day,* it permits any person, whether a British subject or an alien, to locate a mining claim, or to engage in mining in any way, upon the single condition that he shall take out a "free miner's license," for which $5 is paid to the Mining Recorder, and shall maintain the same in force by the subsequent annual payment of $5. (There are other provisions, affecting companies, and providing for larger payments to cover larger terms, which we need not here recite.)

A free miner may locate upon land not already held under the Mineral Act a rectangular claim, not exceeding 1500 feet in length or breadth (horizontal measurement).

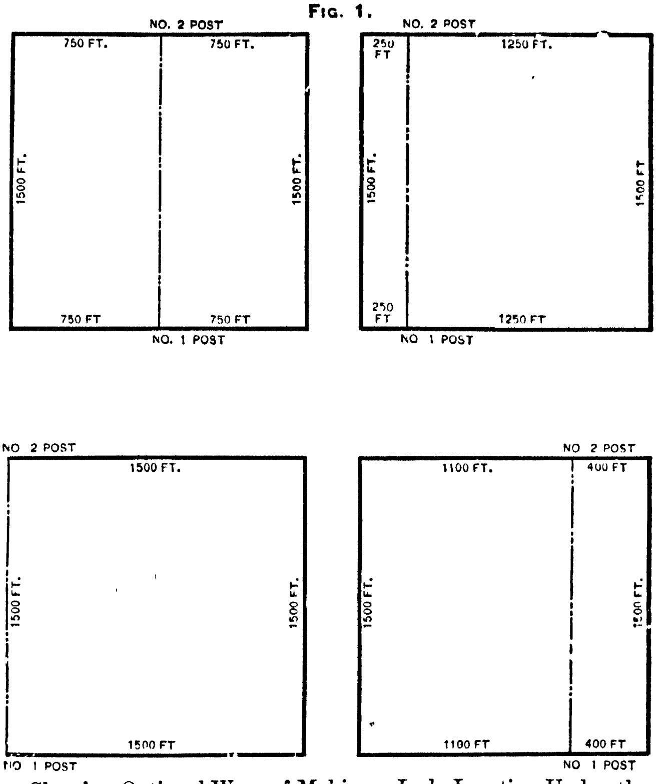
The claim is marked by two posts, No. 1 and No. 2, placed on the boundary, and as nearly as possible on the vein. The distance between these two posts, known as the "location line," must not exceed 1500 feet, and the course must be plainly marked by blazing trees or otherwise. Upon the two posts must be written the name of the claim, the name of the locator, and the date of the location. In addition to the foregoing, upon No. 1 post must be written "Initial Post," the approximate compass bearing to No. 2 post, and the number

* For the full text of the British Columbia Mineral Act, apply to the Minister of Mines, Victoria, B. C.

of feet of the claim lying to the right and left of the "location line." The locator must place a post marked "Discovery Post" where rock has been found in place. The four diagrams of Fig. 1 show the elasticity allowed in locating claims.

Within fifteen days after the location of a claim it must be recorded with the Mining Recorder of the district in which the

FIG. 1.



Diagrams Showing Optional Ways of Making a Lode-Location Under the British Columbian Law.

(The Lode is supposed to run from Post No. 1 to Post No. 2.)

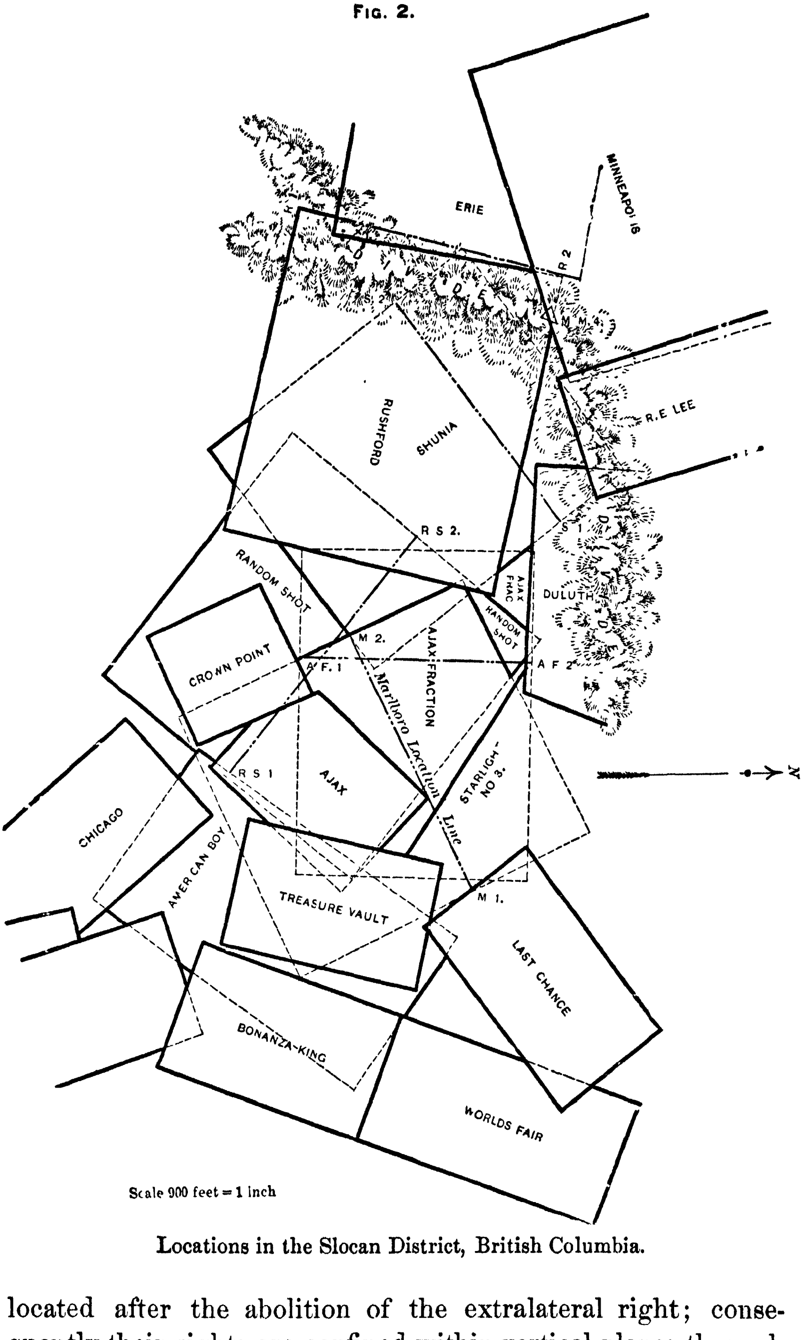
claim is situated. To hold such a claim, after location and record as above, in each year from date of recording, $100 worth of work must be performed and recorded, or in lieu thereof $100 must be deposited with the Mining Recorder. Any free miner or company owning adjoining claims may perform the annual assessment-work for all such claims on any one or more.

After $500 worth of work has been performed (including $100 for the survey of the claim), a "Certificate of Improvements" may be applied for, and after sixty days' advertising, in case no adverse claim is filed, the said certificate of improvements is issued by the Gold Commissioner of the District. Application may then be made to the Gold Commissioner for a Crown grant, which gives a title equivalent to that of the U. S. Patent; such Crown grant being impeachable only upon proof of fraud in connection with the issuance thereof. Surface-rights, for mining purposes only, are conferred by this grant; and all surface-rights, including timber, may be acquired from the government in addition at $5 per acre. If desirable, in securing a certificate of improvements, $500 may be paid to the Gold Commissioner, in lieu of that quantity of work on the claim.

Of course, the change in the law came too late to prevent troublesome consequences. The titles acquired between 1884 and 1892 cannot now be deprived of the vested extralateral right they have secured. As an illustration of the complications thus created, Fig. 2, showing a group of claims in the Slocan district, West Kootenay, B. C., is presented. Of the locations in this group, the Last Chance, World's Fair, Bonanza King, Crown Point, Treasure Vault, Ajax, Chicago and R. E. Lee, were all located prior to the revision of the mining law, and consequently have extralateral rights, the boundaries of which must depend upon mining developments, conflicting testimony, expert opinions, jury verdicts and judicial rulings. After that revision, the American Boy, Starlight No. 3, Duluth, Minneapolis, Erie (location-lines not shown in the drawing), Rushford, Marlboro, Shunia, Random Shot and Ajax Fraction were located in the order, as to date, in which they are here named.

The Marlboro is no longer an existing claim, title having lapsed through failure to perform assessment-work. The Ajax Fraction was located subsequently to cover this ground. The Marlboro having been located before the Shunia and Random Shot, the Ajax Fraction, though a later location than either of these, takes such parts of their claims as conflicted with the Marlboro. This proposition may sound strange to American ears; but the explanation is simple. All these claims were

FIG. 2.



Locations in the Slocan District, British Columbia.

located after the abolition of the extralateral right; consequently their rights are confined within vertical planes through the boundaries of the surface actually appropriated. The

Shunia and Random Shot locations having been, ignorantly or otherwise, so made as to overlap the Marlboro, their locations were entirely void as to the areas thus overlapping; and upon the lapse of the Marlboro title the whole of the Marlboro ground reverted to the public domain, and was open to a new location, unaffected by the imaginary boundaries of earlier claims.

The Rushford, having been located before the Marlboro, takes all the area within its normal boundaries except the parts covered by the Minneapolis and Erie, which are still older. It will be seen that from these conditions no disputes as to title can possibly arise under the mining law which simple surveys and reference to dates upon record cannot settle. The situation presents no greater complexity than a similar division of agricultural land.

Suppose, however, that each of these claims had an extralateral right, like those lying further east in the group. The lode-lines of the Shunia (S 1, S 2), Random Shot (R S 1, R S 2), Ajax Fraction (A F 1, A F 2), etc., indicate lodes of widely different courses, and the resulting extralateral rights would be inextricably confused.

No doubt trouble may hereafter arise in this district, in which some claims were located under the old law and others under the new; but the confusion cannot be as great as if they were all in the former category; and, apart from such local conflicts, the present system will satisfactorily prevent fresh occasions of controversy, and, in new districts, will work out, unhindered, most satisfactory results.

The simplicity of this system is its chief excellence. Difficulties of interpretation are entirely done away. In the complex cases of cross-veins, blind veins, faulted veins, uniting veins, or other irregular deposits, the rights of the miner are clearly defined. The liberality with which he is allowed to make his surface-location, with the vein anywhere within it conformable to one of the location-lines (see Fig. 1), gives him every incentive to a careful preliminary exploration of the real strike and dip of the vein, and rewards his industry and skill in this respect with a grant which goes far to compensate him—indeed, in most cases, more than compensates him—for the loss of the delusive, indefinite and precarious extralateral right.

There is a tendency to perpetual small amendments of the law, which works much temporary annoyance. After nearly every session of the Provincial legislature new rules are promulgated. The unnecessary annoyance of such frequent changes is an evil which should be avoided. So far as possible, changes in a mining code should be thoroughly considered by experts, and being found advisable, should be made final. The experimental manufacture of mining laws is emphatically to be deprecated. This principle should be borne in mind in any revision of the U. S. law, whether radical or partial.

Hübnerite in Arizona.

BY WILLIAM P. BLAKE, TUCSON, ARIZONA.

(Buffalo Meeting, October, 1898.)

THE occurrence of the manganiferous variety of wolframite at a new locality in Arizona was announced in the month of May last.* It occurs in the granite hills of the Dragoon mountains, in Cochise county, about 6 miles north of Dragoon Summit station, on the Southern Pacific Railway of Arizona. It is found in veins of white quartz, traversing a coarsely crystalline granitic gneiss, and, so far as yet determined, is without association with other metallic minerals in the same veins, except small quantities of scheelite. It is a very pure form of the mineral, and contains but little iron, the base being manganese oxide.

The color is brownish-red; it is nearly the same as the color of the mineral originally described by E. Riotte from Mammoth District, Nevada.† Thin films or plates on the broad cleavage-surfaces, and thin cleavage-flakes, seen by transmitted light, are ruby-red, or rather more like the color of red zinc oxide or the mineral known as clintonite. By long-continued weathering the surface becomes less deeply colored, and assumes a bronze-like luster, like the dull oxidized surface of

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* *Eng. and Min. Jour.*, May 21, 1898, vol. lxv., p. 608.

† *Reese River Reveille*, Nevada, 1865; also, H. Credner, in *Berg.- und Hüttenm. Zeitung*, No. 24, p. 370, 1865.

copper. The specific gravity, determined with the proper precautions at 60° F., is 7.140. One cubic foot, therefore, weighs nearly 445 pounds avoirdupois.

The prevailing form of occurrence is in large tabular prismatic blocks, or thick plates, often somewhat radial, penetrating the solid gangue of white quartz. These prismatic and tabular masses vary in size from a fraction of an inch to three, four, or even six inches in diameter; say from one millimeter to two decimeters. These masses of ore are so completely imbedded in quartz that, if there are any crystalline planes or terminations upon the prismatic masses, they are completely hidden in the quartz gangue. The masses cleave easily into tabular blocks. This massive character, and the ease with which the mineral can be separated by culling from the gangue, permit the hand-sorting of considerable quantities for shipment, especially from the numerous weather-worn masses detached from the veins by long-continued weathering. There are, besides, many loose blocks of the hübnerite lying in the soil on either side of the croppings, from which several tons have been already secured for shipment. One mass (containing, however, some quartz) was estimated to weigh 15 to 20 pounds; and masses of equal and perhaps greater weight are not uncommon.*

The veins are of the simplest form or type of quartz-veins. They are nearly vertical, and generally traverse the granitic gneiss in the direction of the rude structural bedding. There is no evidence of movement of the walls upon the veins. The veins vary in their "power," or breadth, from a few inches to two or three feet, and possibly even more in some places. There is much branching and irregularity. While the veins are continuous for considerable distances, in one case for the length of two or three claims, the hübnerite is not so persistent, for there are considerable stretches, along the clearly exposed croppings of white quartz, where no hübnerite is visible. The occurrence is irregular. The mineral is abundant in places for short distances, and then thins out. Considerable work in depth is necessary to show whether the croppings of the min-

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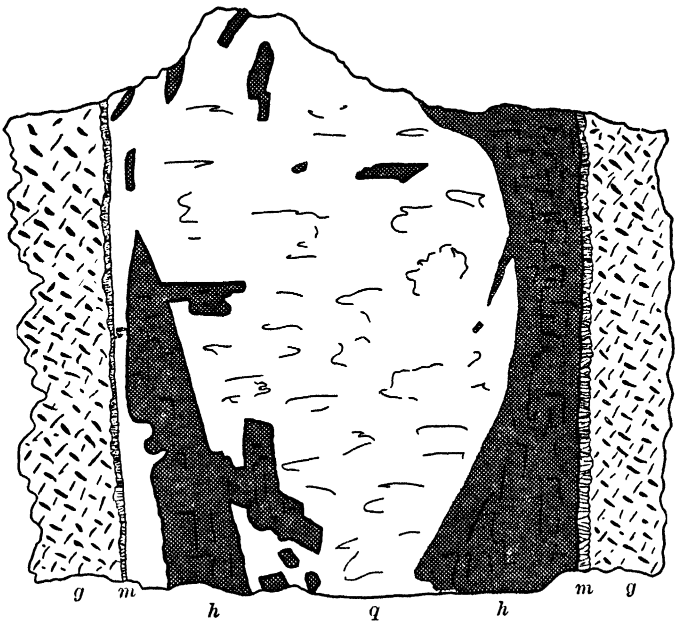
* A mass weighing 500 pounds has been reported since the first manuscript of this paper was sent to the Secretary.

eral are the upward ends or croppings of chutes or chimneys, extending deeply downwards like the chutes upon metal-bearing lodes, or merely lenticular bodies. It can hardly be said that there is any general prevailing form of distribution of the hübnerite in the substance of the vein. It occurs in patches or bunches, sometimes centrally disposed, with quartz on each side, sometimes disseminated from side to side, and again disposed in layers or bunches contiguous to the walls, leaving a

FIG. 1.



g Granite *m* Mica *h* Hübnerite *q* Quartz

Section of a Hübnerite-bearing Quartz Vein. One-half natural size.

central mass of white quartz. One example of this structure is illustrated in Fig. 1, about one-half the size of the original specimen, showing the full width of the vein, with the walls of granite rock on each side.

The medial portion consists of solid white quartz, while the hübnerite is deposited in it upon either side next to the walls, or closely contiguous to them. In this specimen there is a well-defined thin layer or selvage of white mica in small, closely-aggregated plates on each side of the quartz marking the contact-planes, or walls, of the country-rock. These layers

of mica are very thin in places, and have been somewhat exaggerated in the drawing. The folia are disposed at right angles to the plane of the vein. The quartz shows but slight traces of what is known as comb-structure. It is without the well-defined vugs or medial cavities between opposing plates of crystallized quartz, indicative of gradual filling by layers from wall to wall. The whole aspect of the veins is that of the segregated type, rather than of gradual accumulation upon the walls of an open fissure.

As the hübnerite veins are opened in depth, scheelite (calcium tungstate) becomes more frequent. It appears in the high-grade concentrates from the jigs; but the total amount of it is not large, and since it contains at least as much tungstic acid as the hübnerite, its presence is not objectionable.

Purple fluor-spar occurs sparingly, and the mica becomes in depth more abundant, and is more closely associated with the mineral.

Hübnerite compares most favorably with any wolframite as a source of tungsten. It is yet to be shown whether, as an addition to steel, it is not far superior to the ordinary wolframite. Much of the wolframite from other localities, notably that from Cornwall, England, besides its close association with cassiterite, is more or less mixed with sulphides and arsenides, from which noxious elements it must be freed before it can be added to steel. It does not yet appear, so far as I am aware, that the clean manganese tungstate has yet been produced in sufficient quantity, commercially, to establish a distinctive place for itself in steel-making. Its value, as compared with ordinary wolframite, has yet to be studied and made known; and it is hoped that this new locality will afford a considerable supply of hübnerite for years to come.

Since this paper was written, a car-load of the crude ore has been received at the Arizona School of Mines, and concentrated by jigging. The experiment, which has just (November, 1898) been completed, has given excellent results in the form of clean, high-grade concentrates, both from the side-discharge jigs and from those of the Hartz pattern. The tailings carry less than one per cent. of tungstic acid.

Note on the Cost of Tunneling at the Melones Mine, Calaveras County, California.

BY W. C. RALSTON, SAN FRANCISCO, CAL.

(Buffalo Meeting, October, 1898.)

THIS note will give the cost of driving an adit at the Melones mine, in 1898, and, for purposes of comparison, the cost of similar work, in 1888, at the Hogsback mine, Placer county, Cal.

The property of the Melones Consolidated Mining Company is situated on the Mother Lode of California, on Carson Hill, near Robinson's Ferry, and adjoins on the south the famous Morgan mine.

The Mother Lode of California extends for a distance of over 100 miles in length through several counties, with a general strike of northwest and southeast. Three veins (commonly called the east vein, middle vein and west vein) comprise the "lode" in this section.

The Melones property is situated upon the east and middle vein, and includes 1407 feet on the east vein and 5166 feet upon the middle vein.

A shaft had been sunk on the Reserve mine, which is on the east vein, and within 230 feet of the Morgan mine. At the bottom of this shaft drifts and cross-cuts had been run and a large body of ore developed. It was deemed advisable by the Melones Mining Co., which holds this property under bond from the Melones Consolidated Mining Co., to run a tunnel and develop this body of ore at a greater depth.

An option was obtained upon the South Carolina mine, which adjoins the property on the south, and is on the same east vein, upon which a tunnel had been run a distance of 1080 feet. This tunnel was cleaned out and continued to a point within forty feet of the Morgan mine. It passes directly under the Reserve shaft, 425 feet below the 200-foot level.

The additional length of tunnel to be run was 1923 feet, and

cross-cuts and drifts were to be driven from the tunnel to open up the ore. This continuation was made 7 by 8 feet in the clear, with a grade of 3 inches to 100 feet.

Twelve-pound steel rails were used for the track (gauge 22 inches); ties of 4- by 6-inch timber were placed every 3 feet. In most of the tunnel there is a walking-plank 2 inches thick by 20 inches wide.

The air-compressor used in the work was an Ingersoll-Sargent, Class "B," 12¼- by 14-inch piston-inlet, with pulley belted to a main shaft, one end of which carried a 5-foot solid-disk Pelton water-wheel. On the other end of the same shaft was a pulley which belted (when we had no water) to a Nagle engine 12 by 16, Class "A," horizontal. From this shaft was also operated a No. 4½ Baker blower, which was run as an exhaust, using 11-inch pipe.

The Pelton wheel was supplied by a pipe-line 1100 feet long, taking water from the Union Water Company's ditch through 10-inch No. 16 iron pipe for one-half the distance, and 8-inch No. 12 iron pipe at the bottom. All pipe was double-riveted and tarred. This pipe-line gave 200 lbs. effective pressure under a head of 470 feet.

The compressor was started on the 10th day of January, 1898, and machine-drilling commenced.

On the 24th day of September the aggregate length of new tunnel, together with drifts and cross-cuts of the same size, was 2608.5 feet.

During this period of 255 days only two days were lost, and the average progress was 10.22 feet per day, 71.62 feet per week, 306.88 feet per month. Only one heading was driven at a time. Nine sets of timbers were used in the work.

The largest runs made for two consecutive weeks were 92 feet for the week ending May 28th, and 90 feet for the week ending June 4th, or 13.14 and 12.85 feet per day, respectively.*

The rock passed through is greenstone (diabase), brown slate, and heavily mineralized talc schists, filled with quartz stringers. The ground is very firm, as is evidenced by the small amount of timbers used.

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* The largest runs made in 1888, at the Hogsback mine, for two consecutive weeks, were 73.06 for the week ending August 4, 1888, and 66.9 feet for the week ending August 11th, or 10.51 and 9.55 feet per day, respectively.

The tunnel runs diagonally across the strike of the rock, generally at an angle of about 32°; but the strike changes at different points, from square across to parallel with the tunnel.

Owing to the shortness of the bond, early completion of the work was necessary, and no special effort was made to economize in labor or powder.

The working-force, when full-handed, consisted of 29 men, divided into three 8-hour shifts of seven men each (4 machine-men and 3 shovelers on each shift); two drivers with horses working 12 hours; two engineers, working 12 hours; one blacksmith; one helper; one mechanic and one outside man, working ten hours.

The machine-men worked at drilling only; and, to save time, two machine-drills were mounted on one column placed horizontally across the tunnel, as near the top as possible. This permitted the men to place the column and get to work immediately after blasting without waiting for the dirt to be shoveled back from the face; and by the time the shovelers had the rock cleared away, the machines could be swung under the column to put in the center-cut and middle side-holes. After these holes had been drilled, long drills were inserted in the machines, which were then swung to a vertical position, with the drills resting on the bottom of the tunnel; the wedges were then removed from the column, and, the whole weight now being on the drills, the machines were cranked down until the column was in a position low enough to drill the remaining holes. The column was then wedged up, the machines swung into position, and drilling completed. In ground of medium hardness this plan was found very satisfactory; but in hard ground no time was saved. No attempt was made to save powder, as the finer the ground is broken the easier and more quickly it is handled. During the entire work No. 2, 40 per cent., Hercules powder was used, with one stick of No. 1, 60 per cent., in the bottom of each hole. The whole face was blasted at one time—the center cut having a shorter fuse so that it would break first.

In drilling, water was freely used, being conveyed to the machines through 1-inch hose, under 200-pound pressure.

After blasting, the full head of water from the 2-inch water-pipe was turned into the face in the form of a spray, and in

this way the fumes were condensed, the atmosphere was cooled, and the men were enabled to return to work much more quickly.

To save time in changing drills, the two nuts on the U-bolt were riveted, a key-seat was cut in its upper end, and a wedge was inserted, which had a lug on the small end to prevent its falling out. This wedge was put in the U-bolt with the taper-side towards the end of the chuck, so that, after it was driven home, the constant hitting of the drill would keep tightening it. A couple of blows with a hammer loosened it, releasing the drill, thereby saving much time which is ordinarily lost in loosening and tightening the two nuts.

The king-bolts commonly sent out with machines are made too light; and the threads, being fine, are easily stripped. At the Melones, we used king-bolts made of the best 1.5-inch iron, with a No. 4 thread deeply cut, using on this a 2-inch square nut. As a result, no accidents happened by machines falling, which is usually occasioned by the stripping of the threads.

Two Ingersoll Eclipse 3⅛-inch drills (old style) were used, one extra drill being kept in the shop. As an evidence of the efficiency of this old-style machine, it is only necessary to say that for 2608.5 feet of tunnel the total cost of extra parts for three machines amounted to $91.65, and they are to-day in splendid order.

The total cost of repairs and extras for the compressor during the 8.5 months of almost continuous work amounted to $21.32.

The writer having run a similar tunnel at the Hogsback mine in Placer county, ten years ago, believes that a comparison of the records of that work and the work described above will prove of interest.

The Hogsback tunnel ran diagonally across the strike of the rock, which was composed of alternate belts of slate, diorite and quartz. The average hardness was about the same as in the Melones. There was an advantage, however, in blasting, owing to the blocky nature of the ground and the more uniform crossing of the strike, permitting more effective blasting.

The regular force at the Hogsback consisted of 21 men, divided into three shifts of 5 men each, working 8 hours; two engineers, working 12 hours; two drivers and horses, working 12 hours; and two blacksmiths, working 10 hours.

The following tables give the results at each place:

TABLE I.—MELONES.

Actual Cost (Exclusive of Management) of 2608.5 *Feet of Tunnel and Drifts,* 7 *by* 8 *feet, up to September* 24, 1898.

		Cost per running foot.
Labor pay-roll (including timbering), . .	$19,501.46	$7.47
Powder, 2,000 pounds, No. 1, at 16.6 cents, 25,550 " No. 2, " 11.9 cents,	3,405.65	1.30
Fuse, 74,000 feet, at 51.7 cents, . . . Caps, 200 boxes, " 60 cents, . . .	500.20	.19
Wood, 333½ cords, at $5.00, . . .	1.667.50	.63
Water, 15 cents per inch, 40 inches and tender, ...	828.50	.32
Coal, Cumberland, 11,591 pounds, at $15 a ton and freight, ...	179.43	.06
Foot-planks and ties and 9 sets timbers, 8,466 feet at $20 per M., . . .	169.32	.06
Candles, 3,040 pounds, at 7½ cents, . .	262.04	.10
Steel rails, 21,555 pounds, 1¼ cents and 2¾ cents, ...	567.62	.22
Air-pipe, 11-inch, 18 cents and 30 cents, . " 3-inch, 22 cents, ... Water-pipe, 2-inch, 11¼ cents, . . .	1,042.45	.45
Horse feed, hay, 1½ cents; barley, .019 cent,	267.16	.10
Steel, drill-parts, oil, tools, etc., . . .	316.92	.12
Total, ...	$28,708.25	11.02
Actual cost per running foot, . . .		$11.02

The air- and water-pipes used in running different cross-cuts were not left in place, but were moved from one to the other; hence the small cost of this item per foot.

TABLE II.—HOGSBACK.

Actual Cost (Exclusive of Management) of 1559.6 *Feet of Tunnel,* 7 *by* 8 *feet, up to December* 27, 1888.

		Cost per running foot.
Total labor (including timbering), . . .	$12,131.49	$7.77
Powder, 10,021 pounds, No. 2, at 14¾ cents, delivered, ...	1,478.10	.90
Fuse, 23,045 feet, at 54½ cents; caps, 80 cents a box, ...	165.59	.10
Wood, 522 cords, at $2.75, ...	1,435.50	.92
Charcoal, 1580 bushels, at 20 cents, . .	316.00	.20
Candles, 1755 pounds, at 13¼ cents, . .	232.53	.14
Gang-planks and ties, 7624 feet, at $22.50 per M., ...	171.54	.10
Carried forward, ...	$15,930.75	$10.13

Brought forward, ...	$15,930.75	$10.13
Timbers, 21 sets, at $1.80 per set, . . .	37.80	.02
Steel rails, etc., 16 pounds, 20,048 pounds, at 4 cents, ...	801.92	.51
Air- and water-pipes, 3-inch, at 29½ cents, . " " 1 " " 6¼ cents, .	761.43	.48
Horse feed, hay, 3 cents; barley, 3 cents per pound, ...	349.60	.22
Steel drill-parts, oils, tools, etc., ...	916.33	.58
Total, ...	$18,797.83	$11.94
Actual cost per running foot, ...		$11.94

TABLE III.—*Comparison Showing Best Week's Record and Cost of Same at Each Mine.*

	Melones. Report for the week May 28, 1898, 10 A.M.	Hogsback. Report for the week August 4, 1888, 9 A.M.
Holes drilled, . . .	291	150
Holes reblasted, . . .	6	11
Total depth of holes, .	1478 feet	758 feet
Average depth of holes,	5 feet	5 feet
Time used in drilling, .	45 hours 0 minutes	26 hours 0 minutes
Average time per shift,	2 hours 8 minutes	1 hour 15 minutes
Timbers used, . . .	None	None
Powder used, . . .	925 pounds	344 pounds
Candles used, . . .	77 pounds	72 pounds
Water used, ...	40 inches	None
Wood consumed, . . .	None	21 cords
Rock extracted per shift,	23.19 cars	21.8 cars
Rock extracted per day,	69.57 cars	65.42 cars
Total extracted for the week, ...	487 cars	458 cars
No. of working days .	7	7
No. of shifts, ...	21	21
No. of men, ...	31	20
Average progress per shift, ...	4.38 feet	3.5 feet
Average progress per day,	13.14 feet	10.51 feet
Tunnel advanced for the week, ...	92 feet	73.6 feet
Previously reported, .	2310 feet	404.1 feet
Total length to date, .	2402 feet	477.7 feet
Expenses for the week (labor), ...	$551.75	$435.75

In comparing the foregoing tables several differences are apparent.

The small number of carloads extracted per week at the Melones, as compared with the Hogsback, is explained by the

circumstances that the ground at the Hogsback was blocky, and broke higher and wider than was necessary.

The difference in cost of labor per week is explained by the fact that at the Melones, of the 21 miners, only 6 receive $3.00 a day, the balance $2.50 per day, and carmen $2.00; whereas, at the Hogsback all hands were paid $3.00 per day.

It will also be noticed that twice as many holes had to be drilled in the face at the Melones as at the Hogsback mine, because the ground was more difficult to break.

During the progress of the work at the Melones not a single accident of any kind occurred.

To Mr. B. Deleray, the superintendent of the mine, is largely due the credit of the results attained.

Mill Practice of the Utica Mills, Calaveras County, California.

BY W. J. LORING, ANGELS CAMP, CAL.

(Buffalo Meeting, October, 1898.)

It is proposed to describe in this paper as accurately as possible the present practice at the Utica mills, of which I am superintendent. The Utica Company operates three mills, the Madison (40 stamps), the Utica (60 stamps), and the Stickle (60 stamps). My description will be based on the last-named, in which I have experimented more than in either of the other two; but the practice is the same, under the same management, in all three.

Everything is run by water-power, under about 420 feet fall at the Utica mill, and 425 feet at the Stickle mill, giving a pressure of about 185 pounds to the square inch. The Madison mill is situated about three-quarters of a mile below the Utica and Stickle mills, and has only some 100 feet fall from one place, and 38 feet fall from the other. The water that runs the Utica and Stickle mills, and all machinery connected with the corresponding mines, is conveyed through ditch and pipe to the Madison mine and mill, where it develops power for the air-compressor and the 40-stamp mill. The mill is run by a

double turbine, made by the James Leffel Co., Springfield, Ohio. Of course, by using the water repeatedly, as we do, we get our power very cheaply.

As the result of many experiments, I have arrived at a very satisfactory practice in crushing rock at low cost per ton. This object, taken by itself, would simply require us to crush as much rock as possible for each unit of power, plant and time; but we must also keep the loss of gold per ton as low as possible. We have succeeded, I think, in securing both results to a creditable degree.

The rock, as hoisted from the mines, consists of massive quartz, schistose and slaty diabase, with iron sulphurets. At the head of the shaft it is passed over a grizzly made of 3-inch round iron bars, 10 feet long, placed 1¾ inches apart, and set at an inclination of 40°. These bars are supported at each end by a casting made for the purpose, with recesses to receive them. Worn-out and discarded stamp-stems are used, and answer very well.

At the lower end of the grizzly is situated the 10- by 16-inch Blake crusher, which is connected with the grizzly by a sheet-steel hopper, set at the same inclination as the grizzly-bars, and thus feeding the crusher by gravity. One Blake crusher of this size handles all the rock for the 60-stamp mill (aside from that taken out by the grizzly). The amount will be given below. One man on each ten-hour shift furnishes all the labor required at the crusher.

The crusher and grizzly are both set over a bin holding 50 tons, from which the rock is conveyed to three bins in the mill —one for each 20 stamps. The capacity of each bin is about 700 tons. These discharge into "Challenge" ore-feeders, of which one supplies each 5-stamp battery.

The stamps, when newly shod, weigh 835 pounds each, and drop from 7 to 8 inches 100 times per minute. The 10 stamps, in two adjacent batteries, numbered consecutively, drop in the following order: 1, 5, 9, 7, 3, 2, 6, 10, 8, 4.

We keep a duplicate cam-shaft, with cams and pulley, ready to take the place of a broken shaft, which we can thus replace in 3 hours, instead of at least 48 hours, the period otherwise required to strip the broken shaft, turn a new one to fit the cams, and get the ten stamps started again. If a cam-shaft breaks, we take it out, cams and all, and roll in the new

one, with cams already fitted on it. After the two batteries have started again, we can, at our leisure, strip the cams and pulley from the broken shaft, and fit them to a new one, to be ready in case of another break-down. I need hardly say that this arrangement saves much valuable time, as we are using 16 cam-shafts, each operating two 5-stamp batteries, and liable at any moment to break, thereby subtracting 10 stamps from the capacity of the mill, until it can be replaced. The discharge is 10 inches high, three chuck-blocks are used, and as the dies wear the chuck-block is changed, keeping the discharge as nearly uniform as possible. Manganese-steel shoes are used with very satisfactory results. They are 10 inches long, 8½ inches in diameter of face, and weigh when new 177 pounds, and after full use about 28 pounds each. Their life in service averages 296 actual days' work. These shoes are cast with hollow necks, that is, a 2½-inch hole, 3½ inches deep, is cast in the neck, leaving ½ inch of metal around it. This hollow neck reduces the weight about 3 pounds, causing a saving of at least twenty-five cents per shoe in first cost, to say nothing of freight. We have used the manganese steel for both shoes and crusher-plates since November, 1895, and have found it far superior to anything else that we have tried.

The dies are made of hard iron, and last 120 days. They are 5 inches high over all, 8¾ inches diameter of face, and 9⅛ by 9½ inches in base-area. They weigh, when new, 84 pounds, and after using 41 pounds each. They cost 4½ cents per pound as delivered to us, and the old iron is sold to the foundry for 1½ cents per pound.

Round-punched "tin" screens are used; they are made in sheets of 10 by 14 inches. We use No. 1 (equivalent to 30-mesh), which lasts from 15 to 20 days. On clean ore, that is ore free from chips, a screen will last 30 days. These sheets are burned over a mild charcoal-fire before using; just enough heat to burn the tin is all that is required.* The superficial

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* One reason for preferring perforated tin to Russia sheet-iron for screens is the cost. Although the tin-plate is in fact sheet-iron tinned, it is cheaper than the Russia sheet-iron. Tin sheets, 10 by 14 inches in size, cost about $1.35 per dozen, and 3½ to 4 sheets are required for a 5-stamp battery. Russia sheet-iron screens cost from 40 to 75 cents per square foot, which makes them much dearer.

Before using the sheets we burn (that is, oxidize) the tin surface to pre-

area of the discharge is 336 square inches. A splash-board made of $\frac{1}{2}$- by 12-inch clear pine, having the full width of the screen, is suspended to the screen-frame by two eye-straps, riveted to each end of the board, and two hooks, screwed to the screen-frame. A strip of canvas or cloth, 6 inches wide, is tacked to the lower edge of the board, to confine the splash to the apron.

To the mortar below the screen is bolted an iron apron, the bottom of which falls 1 inch below the lip of the mortar, permitting the insertion of a 1- by 12-inch rough board flush with the upper edge of the lip of the mortar. On this the pulp falls from the screen. I have found this board to be a good amalgamator. After a rough board has been used for a month it amalgamates as quickly as a plate, but will not stand the jar like a copper plate. A splash-board used for this purpose can be cleaned in one-eighth the time required to clean a copper plate. This board is protected by a 2- by 6-inch board extending across the apron, and having a $\frac{5}{8}$-inch hole bored in each end to receive two hooks fastened to the battery-posts.

Four inches below the board, on the apron, runs a trough in which are two apertures 3 inches square, delivering the pulp on a copper plate 5 inches wide, with a pitch toward the mortar, whence it passes to the sluice-plates, 2 feet wide, and furnished with 22 feet of $\frac{1}{16}$-inch copper plates set to a grade of 2 inches to the foot.

Of these sluices, each battery has two separate runs, set on independent sets of legs, with wedges to level each sluice independent of the other. Before putting on the plates, the tables are dressed down in the center $\frac{1}{16}$-inch for the full length, causing the pulp to run in angular waves across the plate. If

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vent amalgamation. It is well known that tin will amalgamate as readily as copper or even silver. If a tin screen were used without first burning off the tin, the formation of tin amalgam upon it would soon clog the discharge through it.

Apart from the question of cost, however, and for reasons which I will not here undertake to explain, the tin screens are to be recommended. I have tried Russia sheet-iron, both slotted and punched, and also wire-cloth, and have obtained better results from the tin than from any other screen-material. I think any mill-man who will make trial of it will come to the same conclusion, unless he is dealing with conditions different from those which I have encountered, or can at this time conceive as essentially affecting this part of the mill-process.

the tables are not dressed as described, but are left flat, the pulp will not run so evenly, but will tend to one side or the other. It is a hard matter, at best, to keep the tables perfectly level; and when they have been dressed down in the center, as described, it is much easier to control the flow of pulp.

The head-plates, that is the first 8 feet at the upper end of each sluice, are raw copper; the remaining 14 feet are plated with 2½ ounces of silver to the square foot. We do all our own silver-plating at the electro-plating works owned and operated by the Utica Mining Company.

The plates are cleaned every morning, two men usually working together, and each taking a plate on adjoining batteries. A plug of soft wood is driven into one of the 3-inch holes in the cast-iron apron (described above), causing the pulp to run through the trough and out through the other 3-inch hole on the other plate. The plate they cut off is first washed with clean water, to remove all sand, then sprinkled with quicksilver, and then rubbed with a whisk-broom, to loosen as much amalgam as possible. Every five or six days a weak solution of cyanide of potassium is used in dressing the plates. Great care must be exercised in applying cyanide to amalgamated plates. If too much is used, they will become hard and glassy.

After the plates have been thoroughly treated with the whisk-broom, they are rubbed downward with a piece of 75 per cent. *pure* india-rubber gum, ½-inch thick and 4 by 7 inches in size. The amalgam is then taken up, together with all the sulphurets that may have adhered to the plates during the run of twenty-four hours. The plate is then lightly sprinkled with quicksilver at the head as the case may require, and lightly brushed with the whisk-broom for the full length. The last plate is always brushed upward from the extreme end, so that, in case any amalgam should be hanging to the edge of the plate, it will be brushed up to where it can be readily seen and picked up.

By the methods described two men can dress 24 sluices, 22 feet long and 2 feet wide, in from one and one-half to two hours.

In order to save the sulphurets which are mentioned as having adhered to the plates, and which are very rich (sometimes yielding $10 per *pound!*), the amalgam collected every morning is cleaned in a tank used for this purpose only. At the

end of each month this tank is cleaned out, and the sulphurets are charged, with 25 pounds of quicksilver, into an amalgamating-barrel 40 inches in diameter and 48 inches long, which is run at about 14 revolutions per minute, in which they are treated for eight hours. The charge is taken out through the head (not through the discharging-plug, as ordinarily), with a dipper, and is panned and settled.

The amalgam is cleaned in the usual way, and the sulphurets are sent to the chlorination-works in wooden buckets.

At the lower end of the sluices is a tail-box, having the full width of the two sluices, with a drop of 3½ to 4 inches, and a bottom 5 inches wide. This box has a wing, the full length of the box, with a pitch of about 45° towards the sluice, and extending to within ½ inch of the bottom of the box. The outlet is 1 inch above the bottom of the box. This wing causes the pulp to pass under it and keep the box clear at the bottom, catching the "drip-quicksilver" which is certain to occur in a gold-mill.

From this opening extends a sluice 12 inches wide and 8 feet long, the upper end of which carries ½- by ½-inch riffles, 2 inches apart, while below the riffles are 6 feet of silver-plated copper plates. This box has a grade of ⅜-inch to the foot; and from it the tailings pass to the concentrators, in which the sulphurets are saved.

Three concentrators are employed to each battery of 5 stamps. The Frue vanner, the Union concentrator, and the Tulloch, are all used, and all do fair work.* On each concentrator there is a

* We are using 54 4-foot Frue, 6 4-foot Union, and 16 3-foot 10-inch Tulloch concentrators. From comparative tests of these machines I can find but little difference in the tailings. A special experiment made with the Frue vanner may be of interest to those engaged in milling. The machine has a shaking frame, 12 feet long and 4 feet wide. (Some are 6 feet wide, but we use the 4-foot size.) I dropped the back roller 3 inches by putting a block of wood between the side rail and the bearing that supports the roller; then I took out enough of the small rollers that carry the belt to reduce the total length of the concentrator to 8 feet from center of head roller to center of small roller. From this small roller to the back roller I did not use any rollers in making my test. The belt from the small roller to the back roller had so much grade that nothing was saved on it, so that by this arrangement of the concentrators the effect was just the same as if 4 feet had been cut off from the back end. I had three vanners altered in this way, and made my test by assaying the tailings of these three against that of the 12-foot machines. This experiment continued for 36 days.

The tailings assayed as follows:

sulphuret-discharge, which deposits the sulphurets in a box directly under the head-roller of the machine, and from this box the concentrated sulphurets are taken out every day. The use of this roller does away with the labor of hoeing out the sulphurets. All that is required is to settle the sulphurets and dip out the water, and they are ready to be shoveled out. A car holding 900 pounds of sulphurets is used to convey them to the dump-house. Two samples are taken from each car with a 15-inch sampling-iron; and after all the concentrates for the day have been taken out and sampled, the aggregate sample is sent to the assay-office, together with the aggregate of the samples of the final tailings, taken every three hours. These aggregate samples are assayed every day, and a record of the result is kept. The tailings from the concentrators run through a flume ⅜-mile long, and are worked over a canvas-plant for the fine sulphurets lost in the concentrators. From the dump-house the concentrates are conveyed ¼ mile to the chlorination-works, owned and operated by the Utica Mining Company. These works contain seven roasting-furnaces and a cyanide-plant, the latter being used only on the slimes recovered from the mill-tailings after they have been passed over the Gates canvas-plant, above mentioned.

At the end of a run of from 15 to 30 days, as the grade of the ore may demand, a clean-up of the mill is made in the following manner:

The feed is shut off from 3 batteries (15 stamps) until they have been "pounded out;" then the stamps are hung up, the battery-water is shut off, and two rectangular pans, 15 by 14 inches in area, 3 inches deep on the high side and 2 inches on the low side, are placed in front of each battery, the low side of each being slipped under one of the holes in the apron. These pans are made of the form described, in order to keep the top as nearly level as possible, so that the wash-water will

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	Per ton.
From the altered machine (4 by 8 feet), ...	$0.33
" " unaltered " (4 by 12 feet), ...	0.41

or a saving of 8 cents per ton in favor of the sheet concentrator.

If an 8-foot vanner will do as good work as one 12 feet long, there can be a saving in first cost, and in the cost of excavation and foundation, which is in some localities a large item.

flow all around the top, and not tend to one place, as it would do if the pans were made of uniform height. The amalgam adhering to the screen-frame is taken off with a scraper, made for the purpose, and deposited in a gold-pan; the screen-frame is then removed and thoroughly washed, after which the chuck-block is taken out and placed in the clean-up room to be cleaned; the wooden apron is then scraped, and the amalgam put in a bucket; then the coarse sand in the battery is shoveled out into a box made for the purpose; and then the pan with the cleanings from the screen-frame is put under each stamp, which is thoroughly cleaned of any amalgam that may be found in key-ways, or between the boss-head and shoe. After each stamp has been thoroughly cleaned in this way, the amalgam is taken to the clean-up room and ground with the chuck-block amalgam from the same battery, the weight being kept for each battery separately. The dies are then removed and washed on the apron, and the battery is cleaned out, all the hard sand that has accumulated around the dies being put in the amalgamating-barrel. After the battery has been thoroughly cleaned in this way, about half an inch of sand is put in the bottom of the mortar, the dies are returned to their respective places, the coarse sand is put in, the chuck-block (which by this time has been cleaned) is put in place, the screen is replaced and keyed, and the apron is thoroughly washed (as little water being used as possible), everything being cleaned down into the rectangular pans mentioned above. These pans, and the buckets used around the battery, are washed into the amalgamating-barrel.

By the time the clean-up man has finished the washing of buckets and pans the man who is looking after the self-feeders has started the cleaned-up battery, and has its neighbor ready to be opened. Three men on the batteries and three men at the clean-up room can clean up 12 batteries in three hours.

When the battery is first hung up the little spreader-plates are taken to the clean-up room and cleaned, and the amalgam obtained from these plates is dropped into the tank, which, after the clean-up, is cleaned out, the product being charged into the amalgamating-barrel.

After the mill has been thoroughly cleaned up in this manner, 150 pounds of quicksilver is put into the barrel with water

enough to give a depth of 12 inches over the charge of sand. The barrel is then closed, locked and sealed, and the belt is put on, which drives the barrel at 14 revolutions per minute. This charge is run from thirty to forty-eight hours.

This amalgamating-barrel has a tank of equal capacity built under it, whence a sluice 22 feet long and 12 inches wide, with a grade of $1\frac{1}{2}$ inches to the foot, floored with silvered copper-plates, conveys the pulp and water to either of two settling-tanks, as desired. When the barrel has run its full time it is stopped with the head up, and after the head has been removed a stream of clear water under high pressure is turned on, causing the slimes to come to the top of the barrel, overflow, and fall into the tank beneath. Hence they pass through the sluice into a large tank of 2000 gallons capacity. When all the slime has been washed out the water is stopped, a bucket is set under the $1\frac{3}{4}$-inch plug-hole in the barrel, and the charge is drawn. First comes the quicksilver, which, of course, falls to the bottom of the bucket. A stream of water is kept running through the barrel to wash the sand through the discharging-hole, and the bucket is thoroughly stirred, so that the sand will overflow, leaving the amalgam in the bottom, the "iron" next, and the sand on top. This "iron" is taken out as it accumulates, and after it has all been collected it is screened through a $\frac{1}{8}$-inch screen; the screenings are panned in the clean-up tank, and the result is put back into the amalgamating-barrel at the next regular clean-up.

The amalgam from the barrel is thoroughly cleaned and squeezed through white drillings. The amalgam-balls weigh from 8 to 12 pounds, and yield, when retorted, about 38 per cent. of gold.

The day before the batteries are to be cleaned the accumulations on the sluice-plates are scraped off with steel scrapers, made of old files turned at one end about 2 inches, and ground to a sharp edge. The first 8 feet is scraped in this way, leaving the 14 feet of silver plates to run six months or more without scraping. Great care must be observed in using scrapers on silvered plates, as it takes but little scraping to cut the silver, which not only spoils the plates but makes a very low grade of bullion—a result to be avoided if practicable. The amalgam thus collected is called "accumulations." It is put in a small

amalgamating-barrel 15 inches in diameter and 24 inches long, which is run at 28 revolutions per minute, some quicksilver being added, and enough water to make a thin paste of the charge. The barrel is then sealed up and locked by means of a padlock securing an iron bar which, being itself held firmly by iron lugs on the barrel, covers and completely protects a cap, placed over the discharge-plug.

This charge is run for 24 hours, when it is drawn off in a bucket and cleaned in the usual way, the sulphurets being saved in the cleaning-tank already described. This amalgam yields, when retorted, from 28 to 30 per cent. of gold.

Quicksilver is fed into each battery every hour. The amount of each feed is regulated according to the quality of the ore under treatment, which is judged by the amalgamator from an examination of the board directly under the splash of the battery, where the pulp first falls from the screen. Different ores and different mills require, of course, different practice in this detail of amalgamation.

Every ounce of quicksilver that is fed into the battery is weighed and recorded at the end of each shift, and at the end of the run the amount of the quicksilver thus fed furnishes a practical basis for calculating the probable result of the clean-up before it is made. My own method of making such a preliminary estimate is as follows: Take the total amount of quicksilver fed during the run, and assume a product of $14 to $15 per ounce, and add to this the amount of the "accumulations" mentioned above, assuming a value of $70 gold per pound of accumulations. The total will be very nearly the yield of the clean-up. Of course, these figures would not hold good for all kinds of ore; but if a mill-man is careful he can soon find out, by observation and a few tests, what grade of amalgam he is producing; and the above method, with the employment of figures based on local experience, will be found useful in judging of the probable result before the clean-up is finished, and in detecting abnormal final results which call for investigation.

At many places retorts are lined with chalk, clay, etc. My way is to take good oak ashes, free from dirt; sift them through a 30-mesh screen; make a paste with water, and with this paste line the trays and retort, and thickly seal the head.

The ashes do not shrink, like clay, etc., and the method, which I have practised for many years, is quick, cheap and satisfactory.

The mortars in the mill on which this paper is based are not lined—in fact, were not made for lining. After using them for several years, it was found necessary to do something very quickly, as each end was worn, just above the dies, to $\frac{3}{8}$-inch in thickness, which, of course, would not stand a great while. These stamps were set 10 inches between centers, using $8\frac{1}{2}$-inch shoes, and leaving a space of $1\frac{1}{2}$ inches between each two shoes. The 5-inch dies up to that time had an average life of 59 days, the life of the chrome-steel shoes then used being 191 days. The average number of stamps running per day for 333 days in 1896 was 57 out of the 60, and they crushed 273 tons per day.

In order to line these mortars I found it necessary to reset the stamps, using a guide with $9\frac{1}{4}$ inches between centers, and leaving $\frac{3}{4}$-inch between stamps. In this way $1\frac{1}{2}$ inches was gained at each end of the mortars, allowing room for an end-liner, which served as key to the front- and back-liners. The back was filled with wood carefully fitted in place with the liner outside, saving considerable weight of unnecessary iron. This back-lining is 13 inches high and stands at $77\frac{1}{2}°$, the foot being $1\frac{1}{2}$ inches from the base of the die, which causes the rock, as it is fed into the mortar, to fall on the die. These dies are made with $8\frac{3}{4}$-inch face, while the shoes have but $8\frac{1}{2}$ inches. This I have found to give better results than when both shoe and die are of the same diameter, for the reason that the stem-guides are bound to wear, allowing the stamp to swing. A play of $\frac{1}{8}$-inch in the guide will allow the stamp to swing much more at the shoe, causing the shoe to overhang the die. Now, by having the die cast $\frac{1}{4}$-inch larger than the face of the shoe, we utilize the full crushing surface of the stamp.

On the other hand, if the shoe and die have a greater difference in diameter, the one having the larger diameter will "cup" in wearing, causing a loss in efficiency.

After making the above changes, the mill ran 320 days; average number of stamps per day, 59; average number of tons crushed per day, 296; average life of 5-inch cast-iron dies, 120 days; average life of manganese-steel shoes, 296 days.

The chuck-blocks are made of wood, covered with copper ⅛-inch thick and 8 inches wide, and as long as the battery. In the center of the copper, for the full length, it is bent to fit the wooden block, the upper half being set at an angle of 45°, while the lower half stands vertically. At the lower edge of block, and projecting ¼-inch over the copper, is bolted a ½- by 2-inch iron strap for the full length of the copper. At the bend in the copper, or about 2 inches above the bottom iron, is bolted a second iron strap, ½- by ⅞-inch, for the full length. Six bolts are used, passing through the block and copper, ⅜ by 5 inches in size, with counter-sunk heads. These irons also act as a protection for the amalgam, as it "builds" between them.

I use 8½ gallons of water per battery of 5 stamps per minute, and 2¼ gallons of water per minute on two concentrators, making 10¾ gallons per minute for each battery complete.

The chips and all shoe-wedges—in fact all the wood used about the mill—are saved and burned in a 5- by 8-foot furnace built for the purpose. This furnace has a cement floor, above which are placed grate-bars to the full size of the furnace. The furnace measures, from cement floor to top of bars, 15 inches; from top of bars to top of arch, 3 feet. At the top of the furnace is the charging-hopper, made of cast-iron, with a door to fit. After a charge is burned the ashes are taken out from the ash-box and screened, to remove old nails, etc., that may be in the charge. The ashes are then put in the large amalgamating-barrel with 10 pounds of quicksilver and ground for six hours sharp, as a longer time would cause the charge to "flour," causing a loss. From 6 ounces to 2 pounds of amalgam is saved by this method every month.

The floors all have a slight grade, at the foot of which is a trough leading to tanks which are cleaned out as required, the result being put into the barrel. All cleanings from the mill are put into this barrel and ground for such time as each case may be. It is surprising how much gold can be thus saved in a year by watching the little bits that do not seem to amount to much. Many thousands of dollars are saved around a large plant by such minute care.

For grinding the amalgam recovered from the chuck-blocks I have an iron mortar 12 inches in diameter, the muller of

which has a connection made in such a way that it can be stopped or lifted as desired. This grinding-machine will grind a charge weighing 14 pounds. It has a speed of 65 revolutions per minute.

The labor in the mill is as follows:

One man on each shift of twelve hours, called the feeder, who does nothing but look after the feeding of the batteries.

One night-amalgamator, who attends to the amalgamating and looks after the mill generally.

One concentrator-man on each shift of twelve hours, attending to 36 concentrators.

One day-amalgamator and helper, who attends to the amalgamating and general repairs about the mill.

One mill-superintendent who has charge of everything in connection with these stamp-mills and of the electro-plating works where the plates for the mills are silvered; his time being divided among the mills as required.

Seven men besides the superintendent operate the 60-stamp mill.

The cost of ore milled in 1897, exclusive of power, was 13.8 cents per ton. It is not practicable to give the exact cost of power; but it is, of course, very small, as the water-supply is owned by the company.

The consumption of quicksilver for the same period was 0.076 ounce per ton of ore treated.

Corundum in Ontario.

BY ARCHIBALD BLUE, TORONTO, CANADA.

(Buffalo Meeting, October, 1898.)

JUST one hundred years ago, in a paper read before the Royal Society of London and published in its *Transactions*, Rt. Hon. Charles Greville established and named the mineral species, corundum, the crystalline oxide of aluminum; and we have it on the authority of Professor Judd that in an appendix to Greville's paper the Count de Bournon correctly defined the crystallographic characters of the species. The names of its

gem-varieties, sapphire and ruby, had been in use from a much earlier time; and the name corivindum or corivendum had been given to it by Woodward, in a vaguer way, as early as 1714. In the Burmah corundums every shade of colors from white to the highly prized deep crimson or pigeon's blood are found, and they are named according to colors instead of composition or system of crystallization,—the red variety as oriental ruby, the blue as oriental sapphire, the yellow as oriental topaz, the purple as oriental amethyst, and the green as oriental emerald.

In the western part of Asia Minor and in some islands of the Grecian Archipelago the crystalline limestone which is interbedded with the schists and gneisses carries a blue corundum mixed with magnetite, which is the emery of commerce. The corundum occurs in smaller quantities as a constituent of granite and gneiss in Silesia, Auvergne and elsewhere in Europe; in a compact feldspar rock in Piedmont; in dolomite with tourmaline at St. Gothard ; in crystalline limestone, along with numerous other minerals, in Orange and Westchester counties, New York, and Sussex county, New Jersey, and at various localities in Connecticut, Massachusetts and Pennsylvania. It is said by Dana to be common at many points along a belt extending from Virginia across Western North Carolina and Georgia to Dudleyville, Alabama.

In Burmah, which became a British province in 1886, ruby-mines have been worked for a very long period. There the country-rock is chiefly gneiss, with bands of crystalline limestone of varying thickness and many miles in length. Most of the mining has been carried on in the hill-wash and alluvium carried down from the decomposed summits of hills and mountain ranges; and it has been observed that where the sands and gravels are mixed with a dark, brownish earthy clay, which is a product of the decomposed crystalline limestone, they are richer in such gems as ruby and spinel. The explorations of Barrington Brown appear, indeed, to have satisfactorily established that in Burmah the only rock in which rubies are found in place is crystalline limestone. "It is of the usual composition and character of ordinary crystalline limestones," says Mr. Brown, "being made up of finely crystalline or granular limestone in layers, together with irregu-

larly shaped bands of very coarsely crystalline limestone of white and bluish colors, which are interfoliated with the gneissic rocks." Where a quarry has been worked near Mogok, the matrix of the ruby is a coarsely crystalline semi-opaque limestone of about 20 feet in width. The rubies are found over a space of 6 feet in width, extending almost vertically from the bottom of the quarry to the surface of the ground, and along the center-line, where the rubies are most numerous, are small developments of a grayish diaspore enclosing small crystals of iron pyrites. As to the limestone itself, whether occurring as disseminated crystals through the gneiss or as great interfoliated masses, it is the opinion of Professor Judd that it has been neither organic nor due to direct chemical precipitation in its origin, but has resulted from a metamorphism of the lime-bearing feldspars; while during the process of change from basic feldspar to scapolite, and from scapolite to hydrated aluminum silicates, and from these to aluminum oxide, "the slowly liberated oxide may assume the crystalline form, and thus give rise to corundum." Among other minerals found in the corundiferous limestone are pyrrhotite, hematite, apatite, graphite and spinel.

In Ceylon, in the peninsula of India and in China there are numerous occurrences of corundum in crystalline schists; and in almost every case the mineral is of the gem variety. As far as known to the writer there are no deposits in Asia now exploited for use in the arts, saving the emery of Asia Minor.

In the United States corundum is confined almost wholly to the region of the Appalachian mountains, along a belt that extends from New Jersey to Alabama. In the form of emery it is found at Chester, Massachusetts, in a chlorite belt about 20 feet wide, that lies between formations of hornblende-schist and talc, and traverses the mountains for about 4 miles. There is also a productive emery mine in Westchester county, New York, which ships from 500 to 700 tons of abrasive emery per annum.

Along the Appalachian mountain chain corundum is found in feldspar veins and associated with chlorite in peridotite and serpentine rocks, in amphibolite, dunite and gneiss, as well as in gravel-beds. The principal deposits are found in association

with magnesian rocks, chiefly peridotites, which occur as small lenticular masses in gneiss. As a rule, however, the corundum is neither in the peridotite nor in the gneiss, but in a narrow zone of chloritic minerals between the two. The largest known areas are in the southwestern counties of North Carolina, where corundum was first discovered in 1870. This State has furnished nearly all the corundum of commerce for the United States, but the statistics of the mines and works have never been published. There has been much waste of effort in mining for the gem-varieties, encouraged by occasional discoveries, but chiefly by the attractive colors in which the corundum is found. The whole process of mining and milling has had to be learned by experience; and the task has been made difficult not only by the character of the formations, which is not favorable to sinking or drifting, but also by the closeness with which the corundum crystals adhere to the matrix.*

For abrasive use it is very important that the corundum should be free from particles of rock or mineral softer than itself; and for use as an ore of aluminum it should be free from all impurities, to make extraction practicable by present methods.

The first discovery of corundum in Ontario was made by the late Sterry Hunt, fifty-one years ago, in the second year of his connection with the Geological Survey of Canada. Dr. Hunt explored part of the county of Lanark in 1847. He was joined in some of his excursions by Dr. Wilson, of Perth, who at that time enjoyed some local reputation as a geologist (the mineral wilsonite is named after him), and who is still remembered as a man who paid considerable attention to the natural history of his district. The first place visited by them was the fourth lot on the eighth range of the township of Burgess, upon which Dr. Wilson, a short time before, had discovered a body of apatite. Near by, on the second lot of the ninth range, was a deposit of copper pyrites in crystalline limestone, and this was

* Mr. Alexander Rickard, of New York, who is owner of a corundum property at Energy, in York county, South Carolina, says in a letter to me of recent date: "All our corundums are very difficult to clean. While the gangue is soft it is tough, and adheres to the grains of corundum when it is broken up. This reduces the cutting-value, and also creates trouble by fluxing when making into wheels."

also visited. The only exploration-work consisted of two or three blasts, and among the masses of rock thrown out were some consisting of silvery mica, with quartz, feldspar or albite, and calcspar, holding a delicate emerald-green and almost transparent pyroxene of rare beauty, as well as crystals of a dark honey-yellow sphene. The mica was often aggregated in masses of small crystals having a columnar arrangement,* imbedded in which, and disseminated throughout the rock, were a great number of crystalline grains of a transparent mineral, varying in color from a light rose-red to a deep sapphire-blue. Dr. Hunt, in his report to Sir William Logan, said:

"Their hardness, which is so great as to enable them to scratch readily the face of a crystal of topaz, showed them to be nothing else than the very rare mineral corundum, which from its colors is referable to the varieties known as oriental ruby and sapphire. The grains obtained were small, none, indeed, larger than a pepper corn; but at the time I was on the spot they were not noticed, and the specimens were collected for the pyroxene, in only two or three of which I have since detected the corundum. It is probable that further examinations may develop larger and more available specimens of these rare and costly gems. It is in this crystalline limestone that they generally occur, and the corundum found in the State of New Jersey is in the same rock and with similar mica."

Yet it does not appear that this discovery in Burgess received further attention from Dr. Hunt or other members of the Geological Survey; and the mineral was practically rediscovered there, a year ago, by Professor Miller of the Kingston School of Mining. It will be noticed from Hunt's account that the specimens were collected only for their pyroxene, and that the crystals of corundum were not noticed or identified until a later time.

The largest known deposit of corundum in the Province was discovered twenty-two years ago on the farm of Henry Robil-

* It is not improbable that these were decomposed or altered crystals of corundum. On the metamorphoses of the mineral, Professor Judd says: "At the earth's surface, as is well known, corundum or the crystallized oxide of aluminum is one of the most unalterable of substances. Fragments found in river gravels and sands, though perfectly water-worn, show no trace of chemical alteration in their surfaces. On the other hand, there can be no doubt that conditions must exist in the earth's crust under which chemical change of this mineral does take place; this is abundantly proved by the frequency with which undoubted pseudomorphs of corundum occur. Among the minerals found replacing corundum as pseudomorphs are muscovite (damourite), various forms of spinel, andalusite, fibrolite, cyanite, margarite, chloritoid, zoisite, ripidolite, and other chlorites, various vermiculites, kaolin, and other substances."

lard, in the township of Raglan, Renfrew county; but in this case twenty years elapsed before the mineral was correctly identified. According to Robillard's story, he was returning with his little daughter from a cranberry-marsh on the wide flats of York river, and, in climbing a hill which rises about 500 feet above the river, he sat down upon a large boulder to rest. In telling me the story, Robillard said:

"Annie was kneeling behind me, and picked up a queer-shaped stone, and, showing it to me, said it looked like the stopper of a cruet-bottle. It was just like that; and I wondered what fool of a man had gone to work and whittled it out. Then I looked at the stone where I was sitting; and, bless you, sir, it was paved with cruet-stoppers. And here is the very boulder now," he added, as we reached the spot, about half-way down the hill.

Specimens gathered by Robillard were shown to several persons in Combermere; and one, who professed to be a miner of phosphate of lime in Lanark county, pronounced them to be crystals of that mineral. In 1884 one John Fitzgerald joined with Robillard in an application to the Crown for the mineral rights on the property, including several lots on the 18th and 19th concessions of Raglan; and for a number of years they sought in vain for a customer to buy an apatite-mine. The sturdy pioneers would brook no contradiction of their claim that the mineral was veritable apatite; and when a doubt was raised by two young mineralogists who visited the region about ten years ago in the interest of a capitalist, and a suggestion was meekly made that it might be emery, one of the pioneers cut negotiations short by threatening to "punch their heads." Last year, however, these pioneers were overjoyed to learn, on the authority of an expert, that the mineral was not apatite, but corundum.

Eleven years ago, Professor Coleman, now of the School of Practical Science at Toronto, picked up some boulders of nepheline-syenite in the vicinity of Cobourg, on the shore of Lake Ontario, which held crystals of corundum. A fortnight ago I showed Dr. Coleman several specimens of nepheline, rich in corundum, which I had taken from a large deposit recently discovered in the township of Dungannon, and he at once pronounced them to be identical with his own. "I feel sure now," he said, "that I know where my float-boulders came from."

Twelve years ago, in 1886, Nesbitt T. Armstrong, a farmer and mill-owner in Carlow, discovered corundum on lot 14 in the 14th concession of that township, but he did not know its name, and did not suspect that it possessed any value. A sample was shown to a student of Toronto University, who thought it might be emery; and inquiry stopped there. But in 1893, Mr. W. F. Ferrier, lithologist of the Geological Survey, acquired by purchase a number of specimens collected by Mr. John Stewart, formerly of Ottawa, among which was a package labeled "Pyroxene crystals, south part of Carlow." On examining these specimens, some time afterwards, presumably in 1896, Mr. Ferrier recognized them as corundum, and immediately took steps to ascertain the precise locality from which they came. In October, 1896, he was sent upon this mission by Dr. Dawson, the head of the Canadian Geological Survey, and, guided by Mr. Armstrong, he found the corundum in place upon the lot on which Armstrong's discovery had been made ten years before. Then for the first time the fact was established, on the best authority, that this mineral had been found to exist in Canada in commercial quantity, and that it was valuable as an abrasive material on account of its great hardness. But, as it was too late in the season for field-work, Mr. Ferrier did not extend his explorations beyond that one locality.

The first geological reconnaissance of the district in which corundum has been found was made by the late Alexander Murray, of the Geological Survey, in 1853; but his notes of it are very meager. Mr. Murray made two traverses of the country lying between Georgian bay and Ottawa river—the first from west to east by way of the Muskoka and Petewawa rivers, and the second by way of the Bonnechere and Madawaska to the headwaters of the Trent. The source of the Bonnechere is within a mile of Kaminiskeg lake on the Madawaska, near to where Barry's Bay station on the Ottawa and Parry Sound Railway now stands. Mr. Murray descended the Madawaska to the mouth of its principal tributary, the York branch, or York river; known also, at that time, by its significant Indian name of Shawashkong or Mishawashkong, the river of marshes. The course of this stream, which Mr. Murray ascended, lies for more than 40 miles within the corundum

belt; and along its banks are numerous exposures of syenite, with occurrences of nepheline-syenite. But no reference is made in the report to the rock-formations; and the record of levels for the first 10 miles is of very doubtful accuracy.*

Forty years elapsed before another attempt was made to work out the geology of this interesting area, and the task was then entrusted to the very capable hands of Dr. Frank D. Adams. The area under examination is comprised in sheet 118 of the Ontario series of geological maps, and the four corners of it lie in the townships of Digby, Finlayson, Hagarty and Grimsthorpe respectively, embracing an area of about 3500 square miles. In his first report, made for the season of 1893, Dr. Adams sketched briefly the geological features of the district, the northern portion of which he found to be occupied exclusively with the ancient crystalline rocks of the Laurentian system, and the southern and eastern portions with the limestones and gneisses of the Grenville series. "The discovery of so large an area of the Grenville series in this district," Dr. Adams says in his report, "is most encouraging, as indicating the probable occurrence in it of large and valuable mineral deposits." An extensive and remarkable mass of nepheline-syenite was discovered in the townships of Faraday and Dungannon, which was traced for a distance of over 7 miles in an east and west direction. Dr. Adams says:

> "This is a rare rock, found in but few places in the world, and never before discovered in our Laurentian system. The nepheline is very abundant, forming in many places an almost pure nepheline-rock. The mass is flanked on the south, along a considerable part of its course, by crystalline limestone, and it is also intimately associated with a fine-grained reddish rock resembling aplite. It is of a prevailing gray color, and often has a distinct foliation, coinciding with that of the associated rocks."

The beautiful blue mineral sodalite was also found in a number of places, associated with the nepheline-syenite, in the form of veins and irregular masses; but no occurrence of corundum was observed.

During the past three seasons Mr. Barlow has been associated with Dr. Adams on the work of this field, and a very interesting and valuable report may be confidently looked for

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* The notes of rock-formations and river-levels are much fuller, as well as more accurate, on the large map which accompanied Murray's report.

upon some of the most intricate questions of Archæan geology. Dr. R. W. Ells has also been engaged at intervals in surveying portions of the Ottawa valley east of the area on which Messrs. Adams and Barlow have been working, into which the corundum belt is known to extend as far at least as the Ottawa and Opeongo road. The two map-sheets, however, as well as the accompanying reports, will deal with the general geology of the districts, and notwithstanding the importance of the corundum discovery it is not likely that prominence will be given to that subject, if the usual practice of the survey is followed.

During the last two seasons Professor W. G. Miller of the Kingston School of Mining has been employed by the Ontario Government to make a special report on the field. Beginning last year with a study of the occurrence of the mineral at the place of first discovery in Carlow, he has been able to trace the corundum-bearing rocks eastward across that township, through Raglan and Lyndoch to the shores of Clear lake near the eastern line of Sebastopol, a length of about 30 miles. The breadth of the band varies from half a mile to 3 or 4 miles, and its total area embraces about 60,000 acres. The prevailing country-rock of the district is gneiss, composed chiefly of hornblende, biotite and feldspar, and it is probably an altered gabbro. Numerous dikes or masses, consisting largely of feldspar, cut through the older rocks, which sometimes have the character of coarse syenite, passing in places into nepheline-syenite. In both of these rocks corundum was found, as well as magnetite, pyrite, garnets, zircon and sodalite. In continuing his work this year, Professor Miller has succeeded in tracing the syenite band continuously for about 75 miles, from the township of Glamorgan in Haliburton to the township of Sebastopol in Renfrew, besides tracing it to a considerably greater width over the region exposed last year. Corundum was found at a number of places in the western part of the belt, and a large and apparently rich deposit in a ridge of nepheline-syenite near the middle of it, in the township of Dungannon. But as the rocks, over nearly the whole of their extent, are covered with sand, it is probable that many valuable deposits remain to be discovered. The total area of this band is about 300 square miles; and, as it lies in a Free Grant district, the mineral rights are reserved by the Crown in almost all the lots

that have been taken up for settlement. In a few cases, where lands were sold more than thirty years ago, the mineral rights went with the surface-rights; and since that time some lands have no doubt been sold or leased as mining lands. But it is safe to say that the Crown holds for disposal the minerals in at least 90 per cent. of the whole tract.

Two years ago corundum was found in a property that was being worked as a mica-mine in the township of Methuen, in Peterborough county, about 45 miles southwest from the original discovery in Carlow. This locality has also been explored by Professor Miller this year, and the corundiferous band of syenite has been traced in a northeast and southwest direction about 6 miles, with a width of 2 miles. The range of hills over which it extends is known locally as the Blue mountains, and at its southwest end it reaches the shore of Stony lake.

I spent the last week of September with Professor Miller in going over the more northerly band, from the easterly end of it on Clear lake in Sebastopol to the village of Bancroft on the Hastings road, on the line between Dungannon and Faraday. Only a few of the principal properties were visited, including the Block location in Brudenell, the Robillard location in Raglan, the Armstrong location in Carlow, and a recent discovery in Dungannon, not far from the York river. All these are large deposits, easy of access, and favorably situated for mining operations.

Where the exposure occurs on the Block farm the crystals are in syenite, and are thickly studded in the face of the rock. Outcroppings of nepheline-syenite occur near by; and, owing to its resemblance to limestone, an attempt was made by the owner to burn it for lime. The crystals of corundum have a bronze luster, and vary in size from half an inch to an inch in diameter. Numerous boulders are strewn over the face of the ground which carry a high percentage of the mineral; and in some cases the crystals are nearly pure white in color.

On the Robillard hill, corundum may be traced for a mile or more along its southern face, wherever the syenite is exposed. The corundum crystals are frequently observed to run in strings several inches wide along the surface of the rock, and are of all sizes from half an inch to 2 or 3 inches in diameter, usually barrel-shaped, and ranging from an inch to 4 or 5 inches in

length. On the western shoulder of the hill there is an outcrop of nepheline-syenite; and in this rock the crystals are finely shaped, but of small size—about a third of an inch in diameter and an inch or an inch and a half in length. An expert who has examined this hill estimates the corundum in sight at several millions of tons. There is certainly a large quantity, and in some places it amounts to from 30 to 40 per cent. of the rock mass. Along the foot of the hill are numerous large boulders of syenite, speckled over with crystals like plums in a pudding.

The Robillard hill is cut off by a stream upon its west side from a range of high hills that extends westward 5 miles into Carlow. Professor Miller has carefully examined this range, and has discovered corundum in it at a number of points. The largest showing, however, is on the Armstrong lot, where another stream cuts through, on its way to join York river. The rock has scaled off so as to show a perpendicular face about 300 feet in length and 30 feet in height, exposing a mass of syenite which has been thrust up through the gneiss, and which, in its turn, has been cut by a dike of pegmatite. The gneiss has been thrown up to form an anticlinal arch over the syenite, but is cut through along the north side, where the syenite dike is well exposed with a thickness of 10 or 12 feet. According to Mr. Ferrier, it has been traced along the strike about 700 feet. Crystals of corundum are numerous on the exposed face of the syenite, and are also found in the pegmatite nearest the syenite, which is composed chiefly of feldspar. But where quartz comes in with the feldspar, the corundum disappears. A lot of several tons, taken without selection from this location last year and treated at the Kingston School of Mining, yielded from 12.75 to 15.5 per cent. of corundum.

The last location I examined is in the township of Dungannon. It is in a ridge of nepheline-syenite, having a width of 90 to 100 feet, and rising upon one side to a height of about 60 feet. My time only permitted me to follow it for a length of about 150 yards, but Professor Miller informed me that he had traced it for half a mile. The whole surface, as far as I examined it, was thickly strewn with small crystals of corundum, ranging in color from pearl to blue; but here and there parts of it were altered into white mica. A sample of it, assayed for

me under the directions of Dr. Coleman, carried nearly 10 per cent. of corundum, and was remarkably free from iron. An ore of this character ought to be well suited for the production of aluminum, especially as the nepheline itself, the gangue-rock, contains about 30 per cent. of alumina.

Here it may be remarked that, owing to the presence of iron and other impurities, makers of aluminum assert that native corundum is unsuited for the production of that metal. But it is safer to keep an open mind on problems of this nature. When one reflects that by the adoption of new and improved processes the cost of producing aluminum has been reduced, within forty years, from its weight in gold to 30 cents per pound or less, one ought not to assume that it is impossible to find a process for producing pure corundum at low cost, if not a process to make aluminum out of an impure ore. Professor DeKalb of the Kingston School of Mining was able last winter, with a small experimental plant, to extract corundum 99.61 per cent. pure from rock that carried 5 per cent. of magnetic iron-ore. What, then, might be expected from a large and well-equipped plant, capable of treating 50 or 100 tons per day, supplied with every device that the wit of man can invent, and especially with a good quality of rock to work upon! In one particular, the Ontario mineral appears to differ from the mineral of the Appalachian belt; the gangue is brittle, and is easily broken up and separated from the corundum.

It will certainly add greatly to the value of the corundum-deposits of Ontario if they can be used in producing aluminum as well as the material for abrasives, if the history of that metal during the last ten years is a fair index of its future. In the ten years ending with 1897 its production in the United States has risen from 19,000 pounds, valued at $3.42 per pound, to 4,000,000 pounds, valued at 37½ cents per pound; and so much progress in so short a time seems to be ample justification for the statement of Prof. Richards, made three years ago in the preface to his admirable book on aluminum: "The abundance of aluminum in nature, the purity of its ores, its wonderful lightness and adaptability to numerous purposes, indicate that the goal of the aluminum industry will be reached only when this metal ranks next to iron in its usefulness to mankind."

None of the discoveries hitherto made in Ontario seem to

encourage the hope that gem-varieties of the corundum are to be found, although, in some localities, an occasional crystal is to be seen with qualities not unlike sapphire, being semi-translucent and of bluish color. Perhaps, if search were made in the crystalline limestones, it might be rewarded with better success; not that corundum of any quality has yet been found in the limestones, but because their relations to the gneiss are not dissimilar to those which obtain in Burmah. When the source of the limestones has been worked out, it may be shown that, like those of Burmah, they have been derived by metamorphosis from the feldspar of the gneiss, or perhaps from the feldspar of the syenite; and if so, the analogy would suggest that these rocks are worth prospecting for corundum in some of its more valuable forms. In a note received from Prof. Miller on this subject he says:

"It is quite possible that corundum may yet be found in considerable quantity in crystalline limestone in Ontario, as in India and Burmah. In India the mineral occurs under various conditions in metamorphic (limestones, etc.) and igneous rocks. Of course there need be no connection between the occurrence of the mineral in these two classes of rocks. If corundum occurs in our crystalline limestones, it is of a different origin from that occurring in the igneous rocks (the syenites)."

The crystals discovered by Sterry Hunt in Burgess, it will be remembered, were found in association with pyroxene in crystalline limestone.

In view of the extent and apparent richness of the corundum-fields in the Province, the Government has taken steps aimed at developing the deposits and establishing a home industry. Regulations have been drawn up, under which the mineral rights in lands lying within the two corundiferous belts have been withdrawn from sale, and hereafter the mineral and mining rights in such lands can be acquired only under the leasehold system—the rental for the first year being 60 cents and for subsequent years 15 cents per acre. Instead of allowing speculators to take up and hold lands with a view to sell out their interests to miners and capitalists at a large profit, it is proposed that the advantage of acquiring lands upon the lowest terms shall go to the miner and manufacturer direct; and in the case of parties who will undertake to conduct mining and treating operations on the largest and completest scale, and

who can furnish satisfactory assurance that they possess the requisite capital for the proposed operations (including separation of the ore from its gangue, milling for abrasive uses, manufacture of abrasive goods, and the production of aluminum) the Government may concede a preference in the selection of mineral lands. It is also provided that the Government shall have power to require that all corundum mined from lands leased under the regulations shall undergo certain processes of treatment and milling at works to be erected in the Province to prepare it for market; and may further require, from time to time, as circumstances appear to warrant, that works be established in the Province for the manufacture of all useful or commercial products for which the mineral or ore is economically adapted.

A Description of the Semet-Solvay By-Product Coke-Oven Plant at Ensley, Alabama.

BY WILLIAM HUTTON BLAUVELT, SYRACUSE, N. Y.

(Buffalo Meeting, October, 1898.)

An official of one of the prominent iron companies of the South recently made the following statement during a discussion of the present conditions of the Southern iron business: "The trouble with us in the South has been that we have been satisfied to buy iron and coal lands, build blast-furnaces, and make pig-iron. We stopped at pig-iron, and as a result we are all poor. The South needs diversified industries, and needs to work up its raw products into articles the Southern market can use. If we had realized this sooner, some of us might now be rich."

The South is awakening to this view, and in the Birmingham district alone one basic steel-mill is already in operation, and in a year's time another and much larger one will be turning out rails and shapes, with rod- and wire-mills in prospect.

It is to chronicle another metallurgical step forward in the same district that I have prepared the following description of the new by-product retort coke-oven plant recently put in opera-

tion at the Ensley, Alabama, furnaces of the Tennessee Coal, Iron and Railroad Company (see Figs. 1 and 2).

It may perhaps be well, before describing in detail this particular plant and the availability of its products for the markets of the South, to discuss the by-product oven in general and compare its operation with that of the beehive oven, as many of our members know only the latter, and the difference between the two is very wide, both in construction and in method of operation.

The by-product retort-oven was invented and developed on the continent of Europe, and was specially successful on account of the excellent results it obtained from the lean coals prevalent there, that were coked with difficulty, or not at all, in the beehive oven.

In Europe the retort-oven is used both with and without the saving of the by-products, although most of the recent plants include by-product apparatus.

The retort-oven differs from the beehive in shape, in principle of operation, and in results. As most of us know, the beehive oven is dome-shaped, about 12 feet in diameter and 6 to 7 feet high in the center. The coal is charged through a hole in the center of the roof, and is leveled off in an even layer about 23 inches deep. The fresh charge is fired by the heat remaining in the walls from the previous charge, and the combustion is supported by air admitted through the front door, over the top of the charge. The volatile matter in the coal is driven off by the heat and burned in the top of the oven, along with a portion of the fixed carbon. The source of heat being at the top, the coking extends from above downward, with the formation of long finger-like pieces. The coke is quenched before it is drawn from the oven, thereby preserving the carbon glaze that has been thought so important in the blast-furnace. Thus the process of coking in the beehive oven is effected by the partial combustion of the charge itself.

The retort-oven is a long narrow chamber, from 30 to 33 feet long, about 6 feet high, and from 15 to 20 inches wide, depending on the quality of the coal that is to be coked. The charge is introduced through several openings in the top, and nearly fills the chamber, which is sealed tightly as soon as the surface of the coal is leveled. The ovens are built in blocks

of from twenty-five to thirty, and between each two ovens flues are arranged, in which gas is burned to supply the heat for coking the charge within the ovens.

This gas is a portion of what is driven off from the coal, and by its combustion with hot air a high temperature is readily maintained in the flues. The heat passes through the thin walls of the flues into the coal, which undergoes a true distillation, the volatile matter passing off without coming in contact with any air; consequently no combustion takes place. There is, however, a little breaking down of the hydrocarbons by the action of the heat, and a deposition of carbon that causes the retort-oven to yield a higher percentage of coke than the theoretical, while the beehive oven yields less than the theoretical, owing to the partial combustion of the fixed carbon that takes place.

As the supply of heat in the retort-oven comes from the sides, the flow of the gases generated is from the sides toward the center, while the free expansion of the coke is somewhat checked. As a result, some coals that in a beehive oven make a coke that is too soft and spongy for blast-furnace use, are hardened and strengthened in the retort-oven, and are able to bear the furnace-burden. Owing to the narrow oven and the different application of the heat, the time necessary to coke a charge is much less in the retort than in the beehive oven, the time varying with the coal and the type of retort-oven from eighteen to thirty-six hours for the former, instead of the usual forty-eight and seventy-two hours of the latter.

When the charge in the retort-oven is coked, doors at each end are opened and the charge is pushed out by a steam-ram, and quenched as it leaves the oven. As soon as the ram is withdrawn the doors are closed, and the oven is ready for charging, with practically no loss of heat. The whole operation of discharging and charging an oven can readily be completed in fifteen minutes.

By proper attention to the quenching, it is not difficult to keep the moisture in retort-coke as low as in beehive coke. Its cellular structure causes coke to absorb moisture with great readiness when cold, and a few days' exposure to the air in damp weather will often cause a gain of 10 per cent. of moisture, or even more. It may be well to note here, however, that

the usual laboratory methods for the determination of moisture in coke are quite inaccurate, and the results correspondingly unreliable.

Beehive coke can be produced with less moisture than the most carefully prepared retort-oven coke, for it may for purposes of experiment be left to steam in the oven after quenching until practically dry; but after a few days' exposure to the air, it would not differ in moisture from properly quenched retort-coke.

In the retort-oven without apparatus for saving the by-products, the gases distilled from the coal pass directly into the flues at the sides of the oven, where they are burned, the heat in excess of what is required for coking being used for raising steam. In the by-product plants the gases leave the oven through an opening in the top and enter a collecting-main running along above the ovens. This is usually constructed on the principle of the hydraulic main of the illuminating-gas works. The gas bubbles through water in this main, and part of the tar and ammonia are condensed out as the gas is partially cooled, and are collected and saved.

From this hydraulic main the gas is led to tubular condensers, where it is cooled as thoroughly as possible by contact with a series of tubes through which cold water is flowing. By this second reduction of temperature more tar and ammonia liquor are separated. Then the gas goes to an exhauster, which delivers it to a final scrubbing-apparatus, to remove the last traces of tar and ammonia.

A portion of the gas, thus cooled and purified, is returned to the flues of the ovens, where it is burned, and the remainder, rather less than half, is available for various purposes.

The tar collected from the several operations is pumped into tank-cars ready for shipment, while the ammonia is concentrated into strong crude ammonia liquor, or made into sulphate of ammonia.

The amount of by-products obtained depends greatly on the quality of the coal. A short ton of coal will yield from 15 to 25 pounds of sulphate of ammonia and from 5 to 14 gallons of tar. The amount of gas produced also varies from the same cause, but it is usually from 8,000 to 11,000 cubic feet per ton of coal. 5,000 or 6,000 feet are needed to supply the necessary

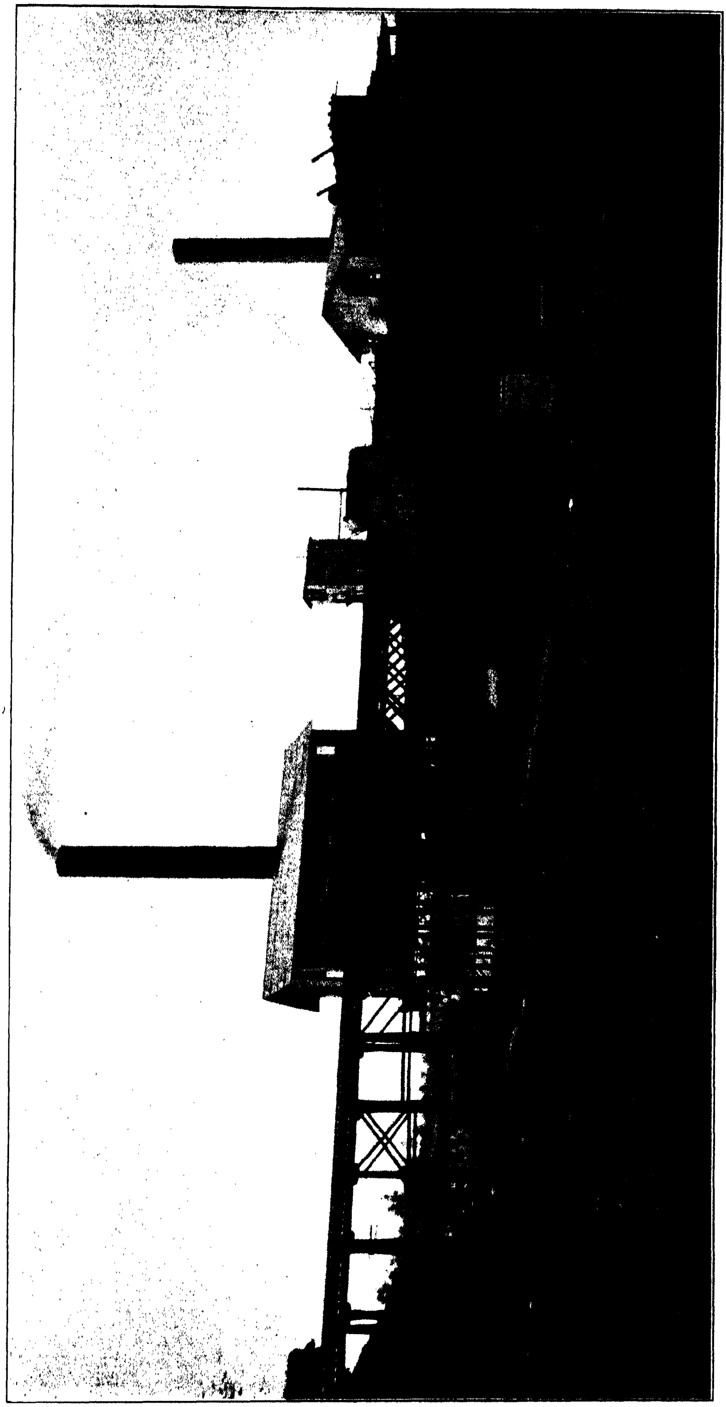
heat to the ovens, the waste heat from which furnishes just about enough steam to operate the plant.

The remainder of the gas is available for any purposes for which natural gas can be used, although it is not so rich. It ordinarily contains from 600 to 700 B.T.U. per cubic foot, while natural gas has about 1000. As ordinarily made, the coke-oven gas is of too low candle-power to permit its use for illuminating purposes, as it averages from 10 to 12 candles. But by taking advantage of the well-known fact that in the distillation of coals, the bulk of the illuminants and hydrocarbons come off in the first part of the operation and the hydrocarbon and carbonic oxide afterward, the distillation can be divided into two parts, a gas of from 16 to 19 candle-power being produced and sold for illuminating-purposes, and the poorer gas used for firing the ovens. A plant of ovens is now in operation which is furnishing without enrichment the whole amount of illuminating-gas used in a city of some forty thousand inhabitants.

While the principles of operation are the same, there are two distinct types of retort-ovens, viz., the vertical- and horizontal-flue types. In the former there are some thirty-odd vertical flues in each wall between the ovens. These are connected at the top and bottom by larger horizontal flues running the length of the oven, the lower one being divided into two parts by a partition midway between the ends. The gas is burned in the lower flue, the flame rising through half the vertical flues and descending through the other half and escaping usually to regenerators of the ordinary reversing type, which heat the air for the combustion. The course of the gases is reversed about every hour, and sent through the flues in the opposite direction.

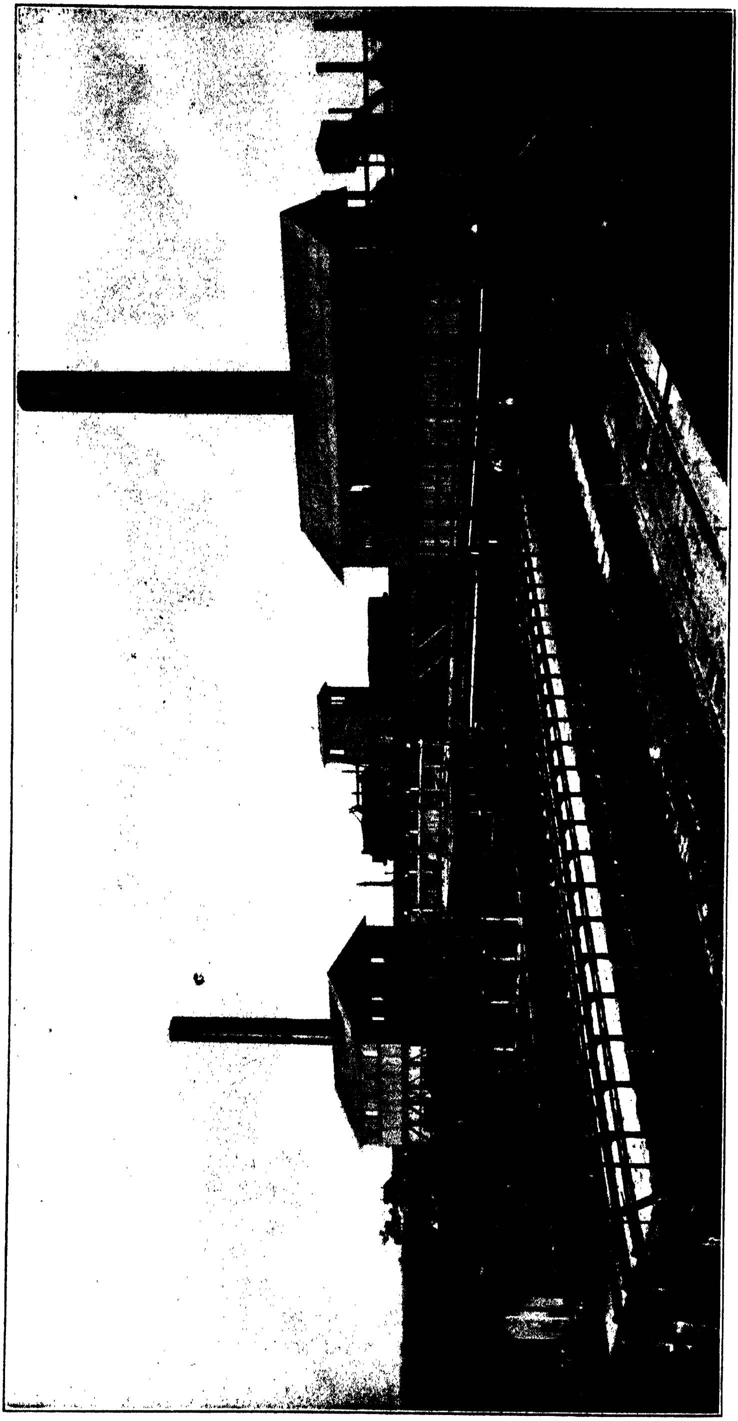
In the horizontal-flue oven the gas is burned in horizontal flues, usually three in number, which are connected at the ends so as to form a continuous system, the gas being admitted through small pipes at the ends of the top and middle flues, where it meets the air for the combustion. The gases travel from above downward, pass under the bottom of the oven, through a simple recuperative arrangement for heating the air, and then to boilers, where steam is made for operating the plant.

FIG. 1.



Semet-Solvay Coke-Oven Plant at Ensley, Alabama.

FIG. 2.



Semet-Solvay Coke-Oven Plant at Ensley, Alabama.

The ovens at the Ensley plant are of the Semet-Solvay horizontal-flue type, illustrated in Figs. 3 and 4.

In this oven the designers have developed the principle that the flue-walls should be thin, to permit the ready passage of the heat to the coal, and that the weight of the top of the oven, which is necessarily thick and heavy to retain the heat, should be removed from the thin, almost white-hot flue walls, and carried independently.

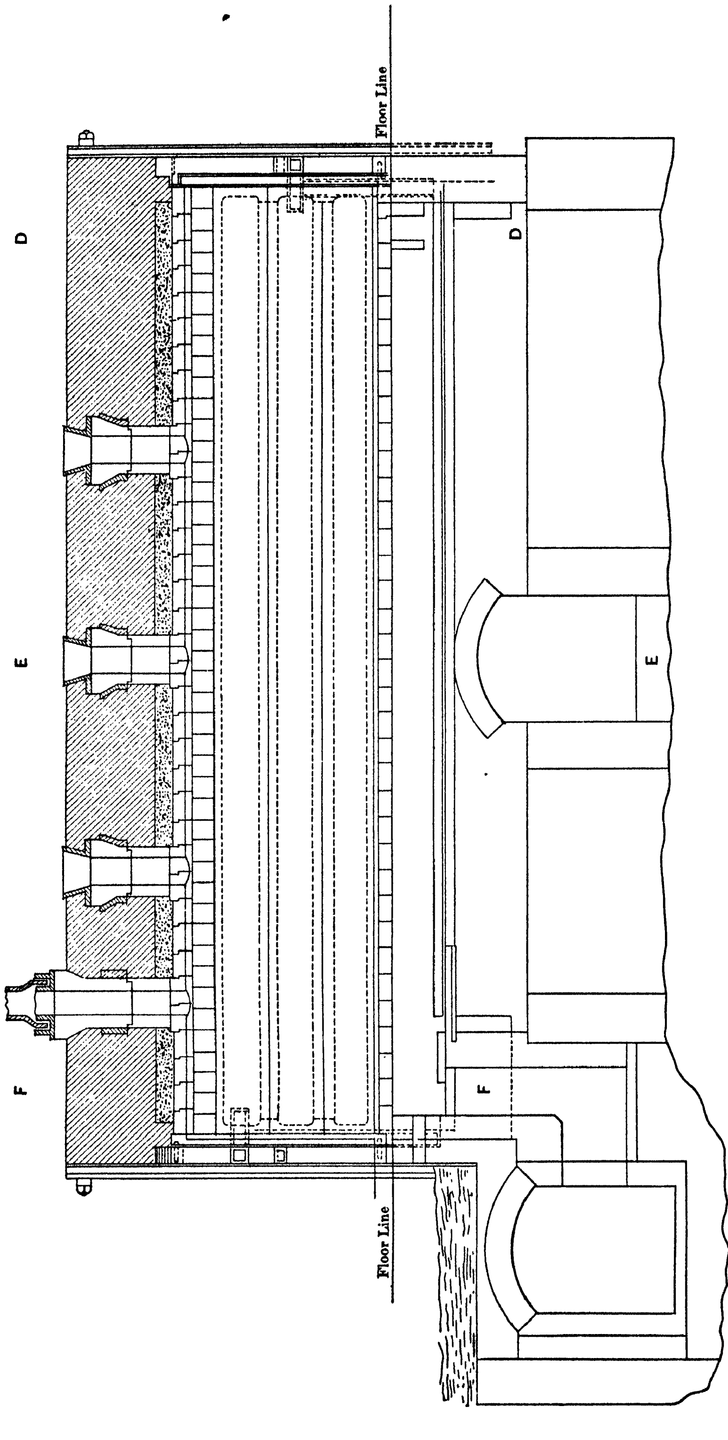
A solid fire-brick wall, 18 inches thick, is therefore placed between each two ovens to carry the load of the roof, coal-cars, tracks and whatever may be placed on it. Each oven, therefore, has its separate set of flues, which may be entirely removed if necessary without affecting the general structure; and, carrying no load, they are free to expand and contract with the changes of temperature incident to the introduction of the cold charge and its heating up to the point necessary to complete the coking-process. Moreover, the thick brick walls form a reservoir of heat that is of considerable assistance in keeping the temperature of the oven uniform. Thus the design of the oven is such as to give a maximum life to the flues (which are the only part of the oven that can wear out), while at the same time it admits their ready repair, should it be necessary, without affecting any other ovens in the block.

The plan of introducing the gas in several places and at the ends of the horizontal flues gives perfect control over the heats in each flue, and permits their examination at any time to see that the proper temperatures are maintained. This arrangement insures that each flue shall have just the temperature best suited to the work to be performed, and prevents one part ot the oven being overheated while another is too cool.

The plant of Semet-Solvay ovens at Ensley consists of 120 ovens, arranged in two parallel blocks of 60 each. The coal used is the washed slack from the Pratt seam, and it is expected that the plant when in full run will produce from 420 to 460 tons of coke per day. This coke will be consumed by the Ensley furnaces.

In the design of the plant careful attention has been given to the problem of handling materials with a minimum of labor, and at the same time elaborate handling machinery has been avoided as being unsuited to the class of labor most available.

FIG. 3.



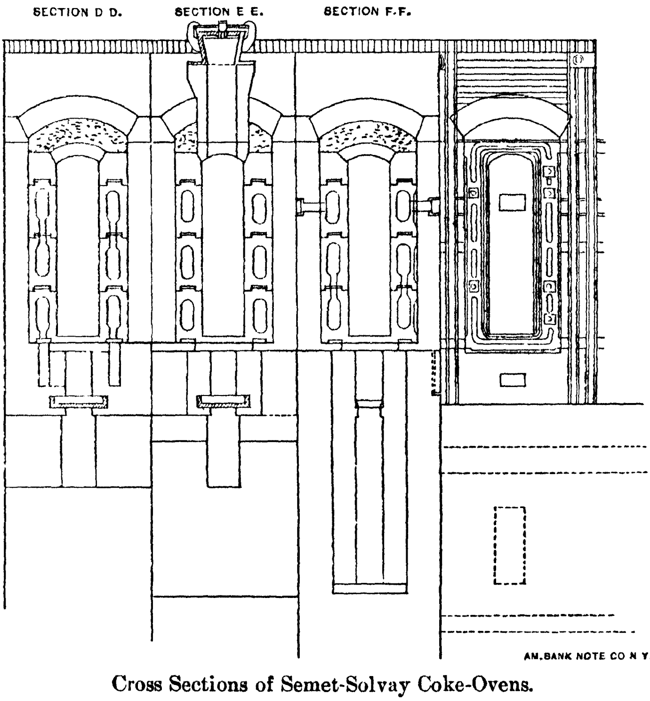
AM. BANK NOTE CO. N.Y.

Longitudinal Section of Semet-Solvay Coke-Ovens.

A spur from the track leading from the Pratt coal-mines and washers is carried directly over the coal-bins above the ovens, the track having a grade at the steepest part of 1.05 feet per hundred, so that a locomotive can deliver a day's supply of coal at one shift.

The coal is delivered in 30-ton cars with sloping hopper-bottoms, and one laborer can easily dump in ten hours the whole day's supply. The bins have a capacity of 1500 tons of coal, to provide against any irregularity in the supply.

FIG. 4.



Cross Sections of Semet-Solvay Coke-Ovens.

The coal for charging the ovens is drawn into larries below the bins and charged into the ovens as above described.

When coked, the charge is pushed from the oven by a steam ram, and quenched as it falls onto a car provided for the purpose. This car is 30 feet long and 7 feet wide, with a sloping bottom, and is so arranged that, when pushed out on it, the coke lies in a thin even layer that permits complete quenching with a minimum resultant moisture in the coke.

The blocks of ovens are located at right angles to the line of

the Ensley furnaces and about 350 feet distant from the stock-house. This arrangement, shown in Fig. 5, permits the most convenient delivery of the coke to the stock-house. By a wire rope and winding-engine the quenching-cars are drawn up a grade of one to six into the stock-house to an elevation sufficient to permit the coke to be dumped into a large bin with a sloping bottom, which in turn discharges directly into the furnace-buggies that are sent to the tunnel-head. Thus the coke is moved but three times after it is quenched until it lies in the furnace, and the breakage is reduced to a minimum. A coke-fork becomes a useless utensil, with a corresponding reduction of labor.

Turning now to the by-products from the coking-plant, the by-product building is located midway between the two blocks of ovens and is of slow-burning mill-construction. Two sets of gas-condensers, one for each thirty ovens, are at each end of the building, and beyond them at one end is the sulphate of ammonia house. Within the building are the necessary exhausters, washers, pumps, tanks, etc., for collecting the by-products.

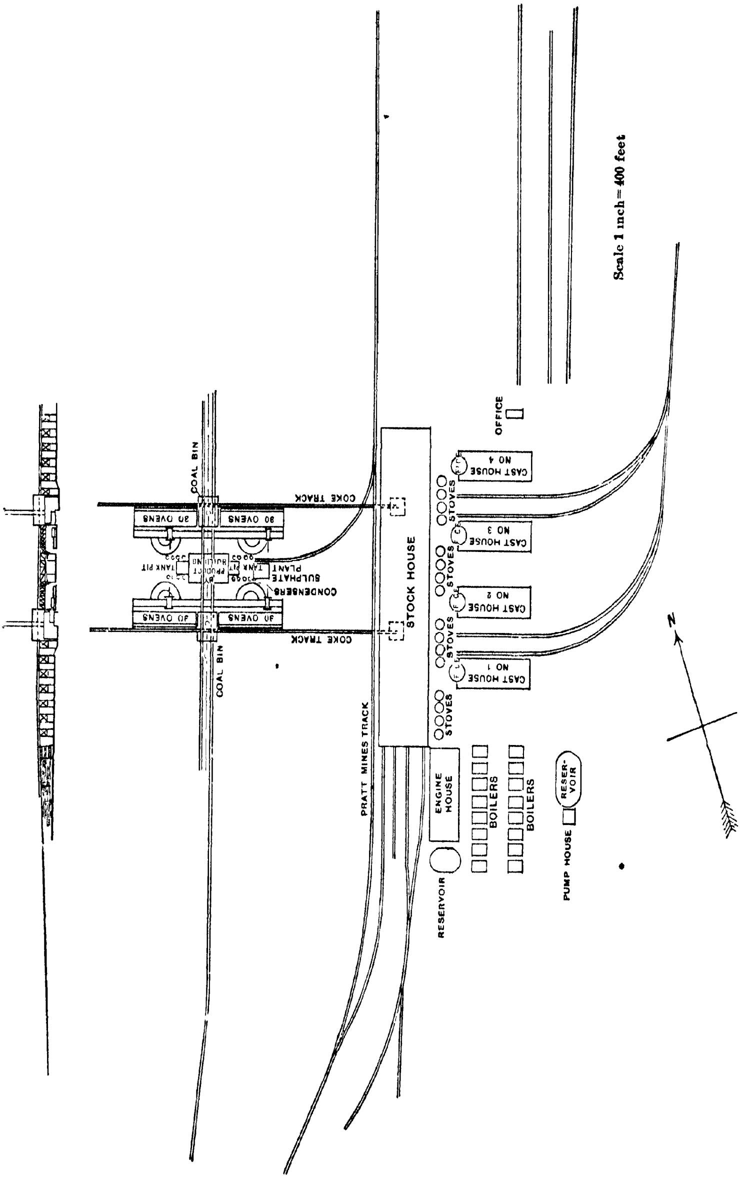
Immediately adjoining the property on the west is a tar-distilling plant, and the tar collected from the gas, after being measured, is pumped directly from the by-product building to its receiving tanks. This plant produces from the tar roofing-pitch, tar-paper and creosoting oils, for all of which products there is a market in the South.

The ammonia produced is at present manufactured into sulphate of ammonia. As the ammonia is condensed out of the gas it is collected in the form of a weak liquor containing probably not more than 1 per cent. of NH_3, along with a good many impurities. To make the sulphate this liquor is distilled with steam and the resulting ammonia gas is absorbed in a bath of sulphuric acid contained in a lead-lined tank. The crystals of sulphate fall to the bottom, whence they are raked out, drained and bagged for shipment.

The sulphate of ammonia finds a ready market, as there are large fertilizer-factories in a number of the Southern States, and sulphate is one of the most valuable sources of nitrogen obtainable. It is worth more per unit of nitrogen than nitrate, blood, or any of the other usual sources of this important element.

Anhydrous ammonia, that is, ammonia gas condensed by compression into the liquid form, is used in large quantities

FIG. 5.



Location Plan of Semet-Solvay Coke Oven Plant at Ensley, Alabama.

throughout the South in the manufacture of artificial ice. This form of ammonia is not produced in the Ensley plant as yet, but it may be arranged for later.

The surplus gas from the plant, of which there will be some two million cubic feet or more per day, is to be used in the new basic open-hearth steel-mill that is being erected within a short distance of the oven-plant. This gas will doubtless prove to be of more value as an auxiliary than as a direct substitute for producer-gas in the open-hearth furnaces. It will be a great convenience for drying ladles, heating soaking-pits, or in other places where comparatively small quantities of gas are required, and of a better quality than ordinary producer-gas. It has been suggested that the open-hearth furnaces be piped for the coke-oven gas, in order to provide an easy means for rapidly controlling the heats, if the occasion should arise.

While the oven-plant is not yet in full run, enough work has been done to make a reliable comparison between the production of coke from this plant and from the beehive ovens of the Birmingham district. It is the practice of some of these plants to coke two 48-hour charges and one 72-hour charge per oven per week. At other places it is more common to run one 72- and one 96-hour charge per week. A comparison of the output of the by-product ovens and the beehive shows that the 120 ovens in the by-product plant will equal in production about 300 beehives making 48- and 72-hour coke, and about 340 beehives running on 72- and 96-hour charges. The charges of coal are heavier when the beehives make only two charges per week.

This difference in the output per oven of the beehive and retort-oven plants is due to the more rapid coking in the retort-oven, although the individual charges are smaller, and also to the increased yield in coke per ton of coal due to the improved method of coking. Tests have shown that this increased yield adds from one-sixth to one-fifth to the amount of coke produced from a ton of Pratt coal.

An estimate of the labor on a beehive plant of 300 ovens compared with that on the plant of 120 by-product ovens as operated at Ensley shows that the latter requires about 15 per cent. more hours labor per day than the former.

For the beehive plant are included only the men required to charge, level, water and draw the ovens and to load the coke onto cars. For the by-product ovens are included all men required about the plant, except the foremen, including those on the ovens and in the by-product plant, delivering the coke into the furnace

stock-house, loading the sulphate of ammonia into cars and delivering the tar to the purchaser. The delivery of the coke is a question of location. If the ovens were not at the furnace-plant the coke would be delivered into railroad-cars.

Notes on the Mines of the Frontino and Bolivia Company, Colombia, S. A.

BY SPENCER CRAGOE, MARIANA, MINAS GERAES, BRAZIL.

(Buffalo Meeting, October, 1898.)

I HAVE read with much interest the elaborate and able paper of Messrs. Granger and Treville on the Mining Districts of Colombia, presented at the Atlantic City Meeting (*ante*, *p.* 33).

Going into detail, however, as the paper does—and particularly with reference to the department of Antioquia, I am somewhat surprised that the property of the Frontino and Bolivia Company, probably one of the richest and most extensive in Colombia, should come off with such scant notice.

The entire property embraces an area of some 28 square miles, and is situated at a distance of about 2 leagues from Remedios—after Medellin probably the most important town of the Department—and lying in approximately 7° N. latitude.

The neighborhood of Remedios has been long celebrated for its production of the precious metal. Alluvial gold in large quantities has been obtained from the vicinity for upwards of two centuries; and even to-day a large proportion of the population finds a comfortable livelihood, during the dry season, in washing the sands of various rivers, notably the Rio Porce; while the amount of gold, in the form of nuggets, brought almost daily to Remedios and other points, from streams in the immediate neighborhood, is astonishing. These nuggets vary much in size. The majority, of course, are small, but larger ones are by no means scarce. I have been considerably surprised at the amount and value of the trade thus transacted between natives and merchants of the various towns.

The property owned by the Frontino and Bolivia Company has been, I believe, for the larger part of the present century,

in the hands of a London company, although up to a quite recent date the development has been hopelessly retarded by the triple drawbacks of great difficulty of access, bad management and insecurity of title.

With the date of its inception and with other matters of historic interest in connection with mining in the vicinity I do not propose to deal, inasmuch as these matters have been already ably covered by a paper published in the *Transactions of the Institution of Mining and Metallurgy* (London), vol. iv. p. 3, by Mr. Frank Owen.

GEOLOGICAL FORMATION.

The formation is granite, bounded on the east by limestone, in which latter, although lodes occur, they are almost uniformly non-productive.

There is apparently no uniform general course of the productive veins, though a direction approximately east and west would conform with one or two of the most important lodes. These, however, are cut through, and in occasional instances slightly thrown, by cross-lodes in almost every direction, all of which, with scarce an exception, are productive.

The ore-bodies are usually small, running from a few inches to perhaps 4 feet in thickness; but, on the other hand, they are often exceedingly rich, and maintain their value fairly well, both along the strike and (to the present date) in depth. Occasionally the veins assume a phenomenal value for a considerable distance.

I recall the main lode of the Marmajito mine, which averaged an assay-value of 30 oz. of fine gold per ton, for a period of several weeks—the lode having a uniform thickness of some 3 feet. In this case, as in some others, the vein-walls were mineralized to such an extent as to render them well worth milling. The lode-stuff in the above instance carried probably 90 per cent. of galena, the remainder being silica and pyrite. In this connection it may be interesting to note that in any ore of this district approaching these phenomenal values, the gold is carried almost exclusively by galena.

Although Colombia is to a great extent remarkable for the variety of ores accompanying its precious metals, yet this is not generally the case with the deposits on this particular property, in which galena and pyrite are by far the main compounds of the base metals. Arsenic (happily for amalgamation) is con-

spicuous by its extreme rarity, if not total absence, up to the present date. I fear, however, that in the Marmajito department there may be considerable trouble from this source as depth is gained.

The general nature of the ore is that of a moderately hard quartz, carrying perhaps an average of 3 per cent. of base metals.

General Features of the Property.

The property is divided into six departments: El Silencio, La Salada, Tigrito, Cordova, Marmajito and Marmajon (naming them from north to south).

El Silencio.—The Silencio mine is worked by an incline put down on the lode; pumping is performed by light pit-work in connection with a large over-shot wheel. The ore is trammed direct to the breaker in the mill-building.

The depth of the mine is approximately some 500 feet upon the lode, or considerably less vertically, the dip being small.

The ore from this mine has been of exceeding richness, and, although there are occasional lapses, it is seldom that there are not "points" that run up to several ounces of gold in assay-value per ton.

The system of mining that prevails generally is overhead stoping, which is done entirely by contract.

La Salada.—This mine, situated about a mile south of the Silencio, is the one department worked by a vertical shaft. The depth is something over 400 feet. Salada is also headquarters, and, among other things, boasts a fitting-shop.

The Silencio main lode runs through this property, but is generally of lower grade than in its more northerly portion. I do not mean, however, that it is at all a "low-grade proposition." On the contrary, there are some very rich "points," and the general average runs over 1 ounce per ton. But the lode-matter carries a smaller percentage of base metals, and, as an inherent consequence, is of lower value, than further north.

Salada, however, has probably the largest reserves of ore opened up in either department. Indeed, for a comparatively high-grade body, the mines are really extensive.

The mine is pumped, like the Silencio, by a large over-shot wheel, and hoisting is performed (as at Silencio also) by a small horizontal engine.

Tigrito.—This property is situated about 2 miles from La Salada, and, like the neighboring mines of Cordova and Marmajito, is worked wholly by adit-levels, driven into the hill.

The contour of the country renders such a system easy, and it is naturally less expensive, particularly as the workings generally are wet.

The mine contains somewhat extensive reserves of mineral. Up to the last year or two it has been comparatively of low grade (7 to 14 dwts. per ton); but lately it has returned very considerable quantities of a high-class rock, and is decidedly improving in its deeper levels.

Cordova.—This property has been richer than Tigrito in the past, and is now improving rapidly, and worthy of very considerable attention. The ore here, though never phenomenally rich, maintains a paying value very evenly, and is perhaps more "free-milling" than elsewhere.

There are several lodes opened up, all of which I believe are now producing. The mine is perhaps more advantageously placed than the others with regard to facilities for opening and drainage, and proximity to the mill.

Marmajito.—This department deserves more attention than is at present bestowed upon it. Apart from the extraordinary value of the ore in some cases (which I have mentioned above), the lode is of a particularly promising nature, and at times massive. The walls are generally well-defined, though in the case of particularly rich "shoots" they give place to more broken country, which, as I have said, is highly mineralized.

Marmajon.—This is merely an extension of Marmajito, but so far the lode has been comparatively poor, and much broken up and faulted, although I hear it has improved.

Since leaving Colombia, I hear that returns have been considerably augmented by some new ground, opened up by another cross-cut, driven beneath the hill in the vicinity. There are also some native workings (on "tribute") along the extension of the Tigrito lode.

MINING COSTS, LABOR AND TRANSPORT.

For the cost of mining, about $5 would be a conservative estimate per ton (2240 pounds), the cost of supplies being very great and the quality of the labor inferior.

One of the largest factors in this cost is timbering, some of the ground being most insecure. Timber is abundant; but, like the quick growth of the tropics generally, it is of poor quality; so that, combining the large quantity required with the necessity of constant renewal, it involves finally no small expense.

Mining, as already observed, is carried out wholly by contract, at prices varying with the class of ground. All measurements are metric.

On the surface, a laborer's wages run from 80 cents to $1, *paper*, per diem (a dollar, paper, is about equivalent to 40 cents gold, at present exchange). Native foremen, carpenters, etc., earn considerably in excess of this. About $60, paper, per month, would be an average.

The company has a roll-call of over 1200 employees, all of whom receive free board. On each department is a large kitchen, or *cocina*, provided with from six to ten native cooks. Meals are served twice daily—beef, bread, frijoles, yams, etc.

Transport.—This is one of the great drawbacks to better working. Goods or machinery shipped from England or the United States are discharged at Savanilla, a small port on the Caribbean, and shipped thence by rail, at a ruinously high rate, to Barranquilla, from which point they are reshipped by river-steamers to a small port, some days up the Magdalena, called Magangue, at the confluence of the Cauca and the Magdalena.* Here they are again transferred to a still smaller craft, that runs, nominally, once a month to Zaragoza, at the confluence of the Cauca with the Nechi. At Zaragoza they are transferred to the mule-train, and conveyed thus inland, about two days' journey. Needless to say, the roads are in anything but good condition.

The natural sequence of these conditions is the entire prohibition of heavy and suitable machinery—a casting in excess of 250 pounds being out of the question—and hence ridiculously inadequate plants at the mines. Of such apparatus and supplies as can be thus obtained, the prime cost is about doubled,

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* Since this paper was written, I learn that the more usual way is to discharge at Carthagena, ship by rail to Calomar, on the Magdalena, and thence by river-boat to Magangue. As an inspection of the map will show, there is not much difference between the two routes. Both involve the same number of re-handlings, the irregular river-service, and the final transportation by mule-trains over bad roads.

to say nothing of the vexation and trouble caused by the delay and losses incurred.

That with all these drawbacks, and with an *average total extraction of not* 50 *per cent.*, the company can pay, regularly, an average net profit of $25,000 gold per month, speaks plainly for the great value of the property.

Reduction.

There are three California mills, "Maria Dama," "San Joaquin" and "Cordova," of 24, 20 and 16 heads, respectively. In addition to these, there are between 60 and 70 heads of native stamps scattered through the property.

Maria Dama.—This mill, situated below the Silencio mine, is driven by two over-shot wheels, the diameter of the larger of which is (I think) 30 feet. The mill is of the old and superseded type of four stamps per mortar, and is in many respects inadequate to its requirements.

The ore passes from a small breaker in the building to bins, and thence to the stamps. The latter are of exceedingly light pattern, barely 400 pounds per head, and are run at well over ninety drops per minute, and occasionally even faster.

Although the mortars are ill-adapted for inside amalgamation, it is still practised successfully—over 40 per cent. of the returns being so obtained. Mercury is fed into the boxes, at intervals of from one to two hours, in quantities relative to the value of the ore under treatment.

The mortar-boxes are cleaned up once weekly, and the plates from once to twice daily, the latter being rubbed up with mercury at intervals of four hours.

The screens used are 40-mesh, diagonal slot-iron. It does not follow that these have proved at all superior to other makes. Unfortunately, the conservatism which pervades the enterprise generally has prevented even the innovation of trial tests on this point. The probabilities, I think, are largely in favor of a 30- or 35-mesh wire screen, were such tests carried out. Considering the character of the ore and the falling weight of the stamps used, the larger discharge-area of the mill-screen should alone give it preference.

No adequate effort is made to maintain an average height of discharge, but it is at no time much over 7 inches. The average drop throughout the three mills is 7.5 inches.

The pulp flows over 12 feet of silvered amalgamated plates. These have the full width of the mortars, and are divided into three sections by drops of about 2 inches. The plates are followed by one 4-foot length of blanketing of similar width. The inclination of the tables is slightly over 1 inch per foot.

From the blankets the pulp is conveyed to Frue vanners, of which there are six, one to each mortar. Upon these a comparatively very poor separation is effected, as a consequence of too much pulp passing over, and ignorant manipulation by the native workmen.

As the amount of stuff crushed is out of proportion to the capacity of the vanners, there was a rude attempt at concentration in round buddles, the "heads" of which were further concentrated in "ties" by native women; but the whole process is (or at least *was*) very inadequate and incomplete, the mill and all in conjunction with it being radically wrong and ill-constructed throughout.

Yet under such obvious disadvantages the monthly output of amalgam averages well over 200 pounds, avoirdupois; and the sulphurets, which perhaps amount to 3 per cent. of the rock crushed, run from 4 ounces to more than ten times that value per ton. Those of them which will more than meet the high freights (over $60, gold, per ton) are shipped to Swansea. The remainder are stored for future treatment, a small percentage being reduced by grinding in crude stone arrastras, with mercury. A considerable proportion washes down the *cañada* to enrich the tailings, which of themselves would be a fortune to some low-grade properties.

The stamps put through an average of 1.65 ton (2240 pounds) per head per 24 hours.

San Joaquin.—This mill lies on the hillside behind Salada shaft, the ore from which is drawn off in cars, from bins at the shaft-mouth, and trammed through a cross-cut to the breaker above the mill. The breaker in question is a Dodge, and is practically useless, its capacity being about sufficient for a 3-stamp prospecting-plant. The greater portion of the rock is broken down by hand-labor to about 2-inch cubes.

The mill itself was erected by Mr. George W. Eustice, the late superintendent, some four or five years since, and reflects

lasting credit on his ability. The battery is of newer and better design, and a trifle heavier, than that at Maria Dama. The stamps, I think, weigh 500 pounds, and run at about 85 drops per minute. The mortars are well adapted for inside plates, which are in use, and discharge on to a similar length of tables as in the previous instance—followed by mercury traps—but no blankets are now in use. The pulp here is again carried to Frue vanners, of which, however, as at Maria Dama, there are too few to be at all compatible with good work.

The power is generated by a 4-foot Pelton wheel, placed a short distance below the mill; and the whole building is lit with electric light from a small dynamo driven from a center-shaft. I might here mention that the manager's house and the other buildings are similarly lighted.

The San Joaquin mill puts through considerably over two tons per stamp in 24 hours.

The percentage of sulphides in the rock milled is somewhat less than at Cordova or Maria Dama.

Cordova.—In the erection of this little mill great credit is due to the assistant manager, Mr. Alfred F. Seccombe, a gentleman of large experience as mill superintendent in various countries. I had the pleasure of working in conjunction with Mr. Seccombe almost from the beginning, and in establishing a small but in many ways model plant from the scantiest of materials.

The mill itself is of a similar type to Maria Dama—being, in fact, the remaining 16 of the original 40 heads sent out for that department.

The mill-frame, battery-posts, blocks and mud-sills were constructed upon the premises, not a few of these being *mahogany!* The ore-bins are of solid masonry, with a storage-capacity of over 400 tons.

The mill is driven by a 3-foot 6-inch Pelton.

Referring to my mill-sheets, I find that the average duty was something above 30 tons per day, passing the ordinary 40-mesh punched screen. The number of drops averaged 100 per minute, and the height of drop was 7.5 inches.

No inside plates were used, but mercury was introduced at intervals into the box—the amount of amalgam so obtained being 40.65 per cent. of the total. The discharge was kept fairly constant at 6 inches.

The discharged pulp flowed over 17 feet of amalgamated copper plates, namely, 4 plates of 48 inches by the full width of the mortar, and one concave "tail-plate" of the same width, and about 12 inches in length.

Thence it passed through mercury-traps and to hydraulic classifiers and settling-boxes in connection with two sets of circular automatic buddles, 18 feet in diameter, which effected a fairly good separation, though, of course, vanners would be preferable.

I have omitted to say that the mills are supplied with "Challenge" feeders, which work excellently.

The "Grizzlies" here are set at an angle of 38°, and spaced a little over 1 inch between the bars; and the breaker (a large Blake-Marsden) is kept fairly uniform at that product—again verifying the greater economy of fine crushing before stamping.

The breaker-power is utilized at night to run the dynamo.

The "heads" from the buddles are reconcentrated in "ties," as at Silencio.

Native Mills.—The native mills, of which there are seven, are composed wholly of wood, with the exception of the square iron shoes and (more rarely) metal dies. They are driven by over-shot wheels (from second gearing) at from 60 to 80 drops per minute, and do much more effective work than might be imagined.

They crush through a 20-mesh punched-screen—the pulp flowing over 12 to 16 feet of blankets, which are washed up in a large tank at intervals.

The tank-sands are rewashed on *bateas* for their gold-contents, and the remaining concentrates are ground with mercury in small arrastras. About 40 per cent. would be an average extraction in these mills.

Miscellaneous.—With the exception of the steam-hoists at El Silencio and La Salada, water-power is exclusively employed. Water is plentiful, but some of it has to come a considerable distance. The company maintains over 30 miles of flume and ditch.

Although transport is high, there is little excuse for the slipshod treatment in vogue. The ore generally is well suited to chlorination, and the process would pay well and cost much less per ton of concentrates than the present shipment-charge

to Swansea. There are many thousand tons of tailings that might be economically cyanided.

Little attention has been practically given to concentration, for which there is a sad lack of plant; and reduction generally has been extremely ill-cared for. I believe Mr. W. George, now in charge of that department, is making strenuous efforts for improvement.

A great drawback is the inability of the present management (which is characterized by good business capacity but ignorance of technical matters) to grasp the importance of anything better than a 50 per cent. extraction, providing that from such an extraction dividends can be paid.

The bullion produced is very impure, averaging .550 fine. The average yield of gold from the amalgam runs from 35 to over 40 per cent.

The gold-dust from the native mills averages perhaps 20 per cent. in impurities, principally galena, which occasions considerable trouble in melting.

In conclusion, it is safe to say that under competent and enlightened management the returns might be increased one-half.

Colombia, as a mining country, is in its infancy, if we measure its age by development and not by simple lapse of time. There is probably no mining-field of richer promise.

Note on the Operation of a Light Mineral Railroad.

BY JAMES DOUGLAS, NEW YORK CITY.

(Buffalo Meeting, October, 1898.)

As the operation of light railroads is important to the mining industry, the following statistics of the Arizona Southeastern Railroad may be of interest.

When the traffic of the Bisbee copper-mines reached 20,000 tons per year the necessity of better transportation than that afforded by eighteen- and twenty-horse and mule teams became imperative on the score of both cheapness and convenience.

Before deciding on the construction of a railroad, the Copper Queen Company made a thorough experiment in transportation

with the traction-engine. A compound engine made by John Fowler & Co. was imported, having cylinders 6½ and 11¼ inches in diameter and 12-inch stroke, constructed to run under a pressure of 140 pounds, and possessing a tank-capacity of 220 gallons. It was put on the road between Fairbank and Bisbee, a distance of about 30 miles, with long grades of 10 per cent. It soon became apparent that this engine was utterly unsuitable for long-distance haulage in an arid region. It could plow its way with difficulty through sand, but after the lightest rain its wheels revolved and it became utterly powerless on a muddy road. If a rain-storm occurred on a clayey stretch of ground the engine came immediately to a standstill. In order, therefore, to secure regular service in all weathers, a hard road-bed was essential. Although the machine was splendidly built, repairs were of course more frequently needed than on an engine running smoothly upon rails, and therefore it could not be used without liability to serious delays at a distance from a machine-shop. The water-supply for such an engine in an arid region also presents considerable complication. Under favorable conditions, however, it hauls very much more economically than animal teams. For some months it picked up the coke-loads of two 18-mule teams (the gross weight of loads and wagons being about 20 tons), hauled them over the Mule Mountain toll-road, up long 10 per cent. grades, and returned daily over the same grades with a load of copper, making a total distance of 18 miles. The same engine is now hauling 30 tons of ore daily, in two trips, from a mine three miles distant from Globe, over a hard, rolling mountain-road, at a cost, for labor, fuel and oil, of 27 cents per ton.

The general use of traction-engines having been abandoned, a railroad to the Copper Queen mine was decided upon, and two routes from Fairbank to Bisbee (a distance in a straight line of under 30 miles, but on a deflected line, around the south of the Mule Mountain pass, of 37 miles) were surveyed. The shorter line with heavy grades and narrow-gauge track was located over the Mule Pass mountain, following substantially the route of the toll-road. The longer line for standard gauge, with 2.5 per cent. maximum grades, was selected. One motive for adopting the longer and more expensive line and the standard gauge was to avoid the transfer of fuel at the junction, and

thus escape the heavy loss in coke involved by every handling, especially when using the friable product of the Colorado ovens.

As a 40-ton locomotive would haul the estimated amount of freight, namely, 30,000 tons annually, in one train daily, and as a 40-pound rail was considered heavy enough to carry safely a locomotive of that size, drawing cars loaded up to the maximum capacity of even 60,000 pounds, it was decided to adopt a 40-pound rail. The rails were made at the Joliet works of the Illinois Steel Co.

The first section of the road, 36.3 miles in length, was built from Fairbank to Bisbee in 1887. This was extended to Benson on the Southern Pacific in the summer of 1895. Most of the rails, therefore, have been in use for ten years. The road runs with easy grades for 30 miles up the valley of the San Pedro, and then commences to climb up long 2.5 per cent. grades. There are 98 curves of an aggregate length of 10.2 miles, the maximum being 12°. Of the 55.3 miles of road there are 45.1 miles of straight track and 10.2 miles of curves. There are 38.1 miles of ascending grade, 10 miles of descending grade, and only 7.2 miles of a level road-bed. The road is therefore exceedingly trying on both track and locomotives, and consequently the 40-pound rails have been put to a severe test. The work they have done is shown in the following table:

Weight and Work of Engines.

Year Ending	Mileage of Loaded Cars.	Mileage of Empty Cars.	Total Tonnage.	Weight of Engines, Tons.
June 30, 1889	34,547	24,634	17,018	36–28
" 1890	65,943	43,536	32,934	36–28
" 1891	68,924	40,088	31,339	36–28
" 1892	73,530	48,474	32,345	36–28
" 1893	76,703	50,022	32,470	36-28
" 1894	80,689	59,982	38,014	36–28
" 1895	138,164	108,110	48,425	36–52–28
" 1896	218,539	152,185	65,905	36–52–28
" 1897	245,116	188,733	83,659	36–52–28–42
" 1898	263,344	156,365	84,739	36–52–28–42
Total	1,265,499	872,129	466,848	

The rails have now been taken up and are being relaid on an extension where light service will be required of them. Only

five have broken during the ten years of use, and to-day they are in perfect condition—neither surface-bent, nor kinked, nor buckled. Only the outside rails which were laid on heavy curves are somewhat worn. The fish-plates and bolts were often found broken, especially in the winter, when expansion and contraction are excessive, owing to the extraordinary diurnal variation of temperature, which often exceeds 60°.

Of course, the preservation of the rails has been secured only by laying them on a well-made gravel and clay road-bed, and bestowing more than ordinary care on the maintenance of the same. Three section-gangs are employed on the 55 miles. The rails were laid on split redwood ties 6 by 6 by 8 feet, 2640 to the mile. These have been cut into by the rails, but show no signs of decay, and, except those on the heavy curves, can be turned over and are good for many years more of light traffic. We are, however, laying our new rails on redwood ties with Servis plates interposed.

Of the four engines used on the light rails, varying in weight from 56 to 28 tons, one, No. 3, is a Baldwin compound engine with four pairs of driving-wheels, a single wheel-truck and 13-ft. 7-in. wheel-base. The high-pressure cylinder is 12 by 24 in., and the low-pressure, 20 by 24 in. in size. The engine weighs, when in working-order, 105,000 lbs., and the weight on the driving-wheels is 89,860 lbs. It has done with equal ease at least 50 per cent. more work than No. 2, a single-expansion Mogul engine, with a single truck and four drivers on a 14-ft. 2-in. wheel-base, the cylinders being 16 by 24 in. in size, and the weight of the engine, in working-order, 76,000 lbs. But extra wear and tear, and the longer period of idleness in the shops, more than compensate for the economy in fuel. In the arid district, where the road is of necessity almost always dusty, and where sand-storms are frequent, locomotives with the fewest wearing-parts are to be preferred.

An objection which we encountered with light rails, when using heavy rolling-stock, was, of course, the unduly rapid cutting of the tires of the driving-wheels.

The change from 40- to 60-pound rails was incident to increased traffic. On the 2.5 per cent. grades a locomotive of safe weight could haul only five cars, carrying an average load and a passenger coach, or the tonnage on which we commenced

operations, of about 30,000 tons a year. To handle, therefore, the increased tonnage indicated in the above table, three engines and three train-gangs were kept in almost constant service, and the operating expenses were reduced, through the larger business, only in the item of maintenance of way.

Economy, therefore, lay in the substitution of heavier rails and heavier locomotives, and a train which would do substantially all the work with one crew.

The point of interest which I think worth recording is the amount of work done by the lighter rails, and the adaptability, therefore, of a rail of that weight to branch-roads of moderate traffic. It must be confessed, however, that to-day, with the wonderful diminution in the cost of rails, the inducement to economize in that item of construction is not so great as formerly.

Note on Slips and Explosions in the Blast-Furnace.

BY F. B. RICHARDS, CLEVELAND, OHIO.

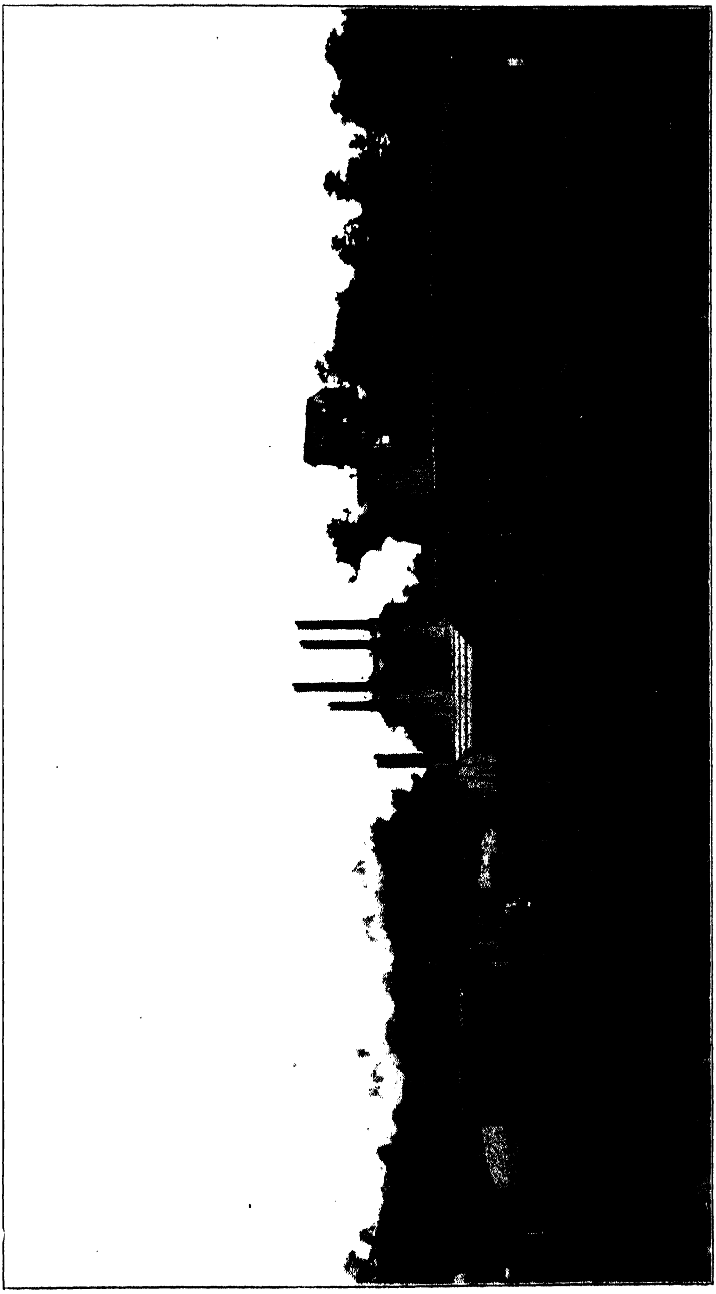
(Buffalo Meeting, October, 1898.)

FOR the last five years the furnace-men drawing their ore-supply from the Lake Superior districts have had to solve the problem of using the very fine Mesabi ores as part of their ore-mixture.

It is not my purpose to dwell on the early experiences and the progressive development in the use of these ores up to this time, but to lay before the members of the Institute one prominent feature concerning them, namely, the frequent and sometimes very disastrous slips (?) or explosions (?) occurring at furnaces where these ores are used.

The *Transactions* of the Institute contain very little on this subject. Mr. F. E. Bachman, in his discussion of Mr. O. O. Laudig's paper (*Trans.*, xxvi., 1061), advances a theory as to the cause of these occurrences; but this is the only reference to the subject in the *Transactions* up to this time, so far as I can recall. Probably all the furnace-men running the hard-driven coke-furnaces in the central West have had more or less experience with this trouble; and I believe the majority of

FIG. 1.



The Claire Furnace, in Normal Condition.

FIG. 2.

The Claire Furnace, Five Seconds after the Explosion of September 15, 1898.

them will agree that a decided explosion takes place, while others feel that it is a heavy slip, followed by a rush of gas. Without citing evidence gathered in many cases, I wish to submit one instance of such an explosion, which is illustrated in Figs. 1 and 2, reproducing photographs. Fig. 1 shows the Claire furnace at Sharpsville, Pa., in normal condition. Fig. 2 is from a photograph, taken about five seconds after the explosion. The circumstance that the photographer happened to be, at that particular moment, in the proper position and in the necessary condition of preparation to take this picture immediately upon the unexpected occurrence, and to get an excellent view of the appearance it presented, is a coincidence not likely to occur often. It need scarcely be observed that furnace-managers are not able to foresee accidents of this kind, so as to have artists on hand at the critical moment.

The Claire furnace is 75 feet high by 16 feet bosh-diameter and 10 feet 6 inches hearth-diameter, and is working with 13 tuyeres on an ore-mixture containing 37.5 per cent. of a Mesabi ore, 25 per cent. of which will go through a 100-mesh sieve, the balance of the mixture being Menominee and Marquette range hematites. On September 10, 1898, there had been two explosions while working 50 per cent. of Mesabi ore in the mixture; hence it had been reduced to 37.5 per cent., as above stated. As early as 6 A.M. on September 15th the furnace commenced making light slips, which continued at intervals of about 30 minutes up to casting-time, 10.30 A.M. After starting up again after the cast, the slipping continued, the stock settling so hard at times that one standing beside the columns could feel the jar. The red ore-dust showed continuously at the chimneys of the boilers and stoves, indicating an irregular settling of the stock. The slipping had been so extreme that the coke and melted cinder had packed tight in the hearth, and no cinder could be got at flushing-time; it came back into the blow-pipes with succeeding slips, but was forced back again by the blast. There are on the furnace, directly under the platform, two explosion-doors 6 feet in diameter; but up to this time none of the slips had opened either of them. At 1.30 P.M. there was a terrific explosion, and both explosion-doors opened to their full capacity, hurling ore, coke, scrap and limestone for several hundred feet. A piece of limestone 6 inches in diameter was picked up 425 feet away from the furnace.

Fig. 2, taken, as explained, by chance, shows the furnace about five seconds after the explosion. The great cloud was composed of ore-dust, gas and finely-divided carbon. The latter was in the form of lamp-black, and, as can be seen, was of considerable volume.

No damage was done, except to the windows of the office and laboratory, and the roof of the blacksmith-shop. The hopper was fastened down; otherwise it would probably have been thrown out, as has happened in other cases of this kind. Operations were resumed at once; cinder was drawn in the course of the afternoon; and the furnace cast 33 tons of white iron at 6 P.M., and on the following cast made the usual quantity of good gray iron.

It is my purpose, in submitting the above, to arouse some interest on this subject on the part of furnace-managers, in the hope that suggestions of theory as to the cause of such occurrences, and of practice as to the remedy, may be elicited.

I am indebted to Mr. J. W. Robbins, Superintendent of the Claire furnace, for data of conditions before and after the explosion, and to Mr. A. M. Robbins, General Manager of the Claire Furnace Company, Limited, for the photographs.

Analysis of Blast-Furnace Gas While Blowing In.

BY RALPH H. SWEETSER, EVERETT, PA.

(Buffalo Meeting, October, 1898.)

WHEN a furnace-manager is "blowing in," he generally has no time to consider the composition of the waste gas, and does not bother with it, except to take care that he does not get "gassed." Moreover, the analysis of the gases while the wood is still burning has not much to do with the economy of the normal working of the furnace; yet, on account of the deadly effects of the "blowing-in" gas when inhaled, and of its high explosive qualities, it is interesting to know its composition. In the *Transactions* there is no record[1] of the analysis of the gas during an up-to-date "blowing-in." Even in Mr. Whiting's admirable paper (*Trans.*, xx., 280) the reported analyses did

not begin until ten hours after the furnace was lighted. In that case, however, the blast was not put on until fifteen hours after the fire was kindled.

Through the courtesy of Messrs. David Baker and A. K. Reese, the writer had, in May, 1897, an excellent opportunity to take up this matter and analyze the "blow-in gas." At that time "C" furnace of the Maryland Steel Co., at Sparrow's Point, Md., was being blown in for its second blast. The furnace was dried out and then filled with wood and stock, in the manner usually followed at the Maryland Steel Co.'s furnaces. Pine cord-sticks were laid flat in layers, crossing alternately, till the hearth was filled solidly up to the tuyeres. Smaller kindling-wood was filled in at the tuyere-level, and then cord-sticks were laid flat in layers up to about 6 feet above the tuyeres. The coke- and ore-charges were then dumped in until the furnace was full. The hot-blast ovens were fired with wood for several days.

On Wednesday morning, May 26, 1897, the down-comer and dust-catcher were fired with wood, and the tuyeres were filled with cotton waste soaked in coal-oil. A wood fire was lighted on the bell at 11.40 A.M., and then everything was ready to blow in.

The furnace was lighted at 11.55 A.M. by thrusting red-hot iron rods into each of the sixteen tuyeres, just as the blast was put on. There was a slight gas-explosion, not more than a puff, at the top, when the gas reached the fire on the bell (which was open). The bell was closed at 12.23 P.M., and the gas put into oven No. 1. The first gas sample was taken at 12.26 P.M., thirty-one minutes after the furnace was lighted, and only three minutes after the bell was closed.

All gas-samples were taken at the bleeder, a few feet above the floor of the tunnel-head, and were drawn through a perforated iron pipe, introduced through a hole in the side of the bleeder. The samples were taken and analyzed in an Orsat apparatus by the writer as fast as was possible under the circumstances. The results represent percentages by volume (see table on page 610).

These and other data are graphically shown in the diagram, Fig. 1.

At 1.25 P.M., May 26th, there was not much gas burning

Analysis of Blowing-in Gas.

Time.	Revolution Engine.	Blast Pressure. Lbs.	Blast Temp. Fahr.	Stock.	CO_2. Per Cent.	CO. Per Cent.	O. Per Cent.	CH_4. Per Cent.	H. Per Cent.	Total. Per Cent.	$\frac{CO}{CO_2}$.
May 26.											
12.26 P.M....	15	0	300°	About full.	10.8	16.2	0.6	1.1	6 66	35.36	1.50
1.33 "	15	0	320°	"	3.2	27.0	0.0	4.0	2.10	36.3	8.44
2.44 "	18	0.8	450°	"	2.8	28.8	0.0	4.2	0.0	35.8	10.28
4.05 "	18	1.0	500°	"	3.0	28.6	0.0	4.2	0.0	35.8	9.53
5.13 "	24	1.6	425°	"	3.6	28.4	0.0	...	...	...	7.88
9.25 "	30	1.75	620°	Out of reach.	4.6	28.0	0.0	...	...	...	6.08
10.45 "	30	2.25	450°	"	5.8	28.4	0.0	...	...	...	4.89
May 27.											
11.00 A M....	28	2.50	730°	"	4.5	...	0.1	...	...	...	...
2.45 P.M....	28	3.75	860°	"	5.0	26.8	0.0	7.4	0 0	39.2	5.26

around the furnace, except at the tap-hole, which had been left open. The flame at the tap-hole burned wildly until 3.15 P.M., when the hole was shut with some difficulty. About this time the gas burned fiercely, and would explode with much force. It is very important that the tap-hole should be shut without taking the blast off; for the gas at this stage of the blowing-in explodes very readily and with much force, and an explosion in the down-comer is likely to cause an explosion throughout the whole furnace. It is essential, therefore, that the blast be kept on continually until after the gases become less explosive. It will be seen in the table that at about this time (2.44 P.M.) the amount of CO_2 was at a minimum, and that of CO at a maximum; and the ratio $\frac{CO}{CO_2}$ was highest.

At 4.05 P.M., May 26th, the stock had settled a little on top, but it did not move much more until 6.22 P.M., when it suddenly dropped "out of reach,"* and remained "out of reach" until 10 A.M. the next day. If the furnace had been kept full the percentage of the gases would doubtless have varied regularly, and the ratio $\frac{CO}{CO_2}$ would have decreased steadily instead of increasing again at 2.45 P.M.

At 3.05 A.M., May 27th, cinder appeared at tuyere No. 1, and the first flush took place at 3.25 A.M. The first cast of iron was made at 6.15 A.M., and gave good grey iron, with 1.40 per cent. of silicon and .011 per cent. of sulphur.

* That is, so far that its surface could not be reached by the iron rods, about 18 ft. long, ordinarily thrust down vertically from the tunnel-head, to determine the location of the top of the stock in the furnace.

Up to 5.13 P.M., May 26th, when the fifth gas-sample was taken, the smoke at the top of the draft stack was not very dense, but when the sixth sample was taken at 9.25 P.M., the

FIG. 1.

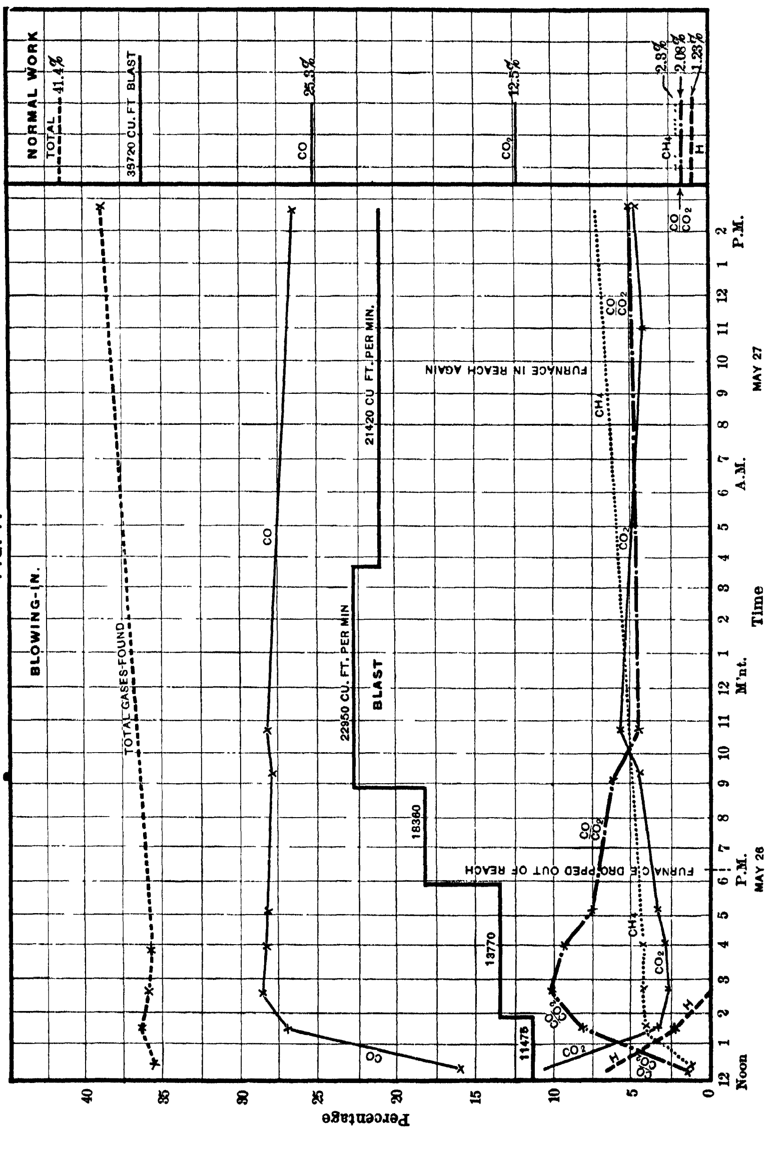


DIAGRAM SHOWING GAS-ANALYSIS DURING BLOWING-IN, AS COMPARED WITH NORMAL FURNACE-WORK.

smoke was white and dense, and continued so during the night and next day.

In examining the diagram, Fig. 1, it will be noticed that the

gases change greatly and rapidly during the first few hours. During the second and third hours the waste gas is made up mostly of carbonic oxide with considerable hydrogen and marsh gas; there is but little carbonic dioxide, and the ratio $\frac{CO}{CO_2}$ is at its height. The waste gas is now dangerous if not handled properly. If breathed, its effects are quick and violent, and sometimes fatal. If ignited while confined within the dust-catcher or gas-mains the explosion is instantaneous and destructive. In no case should the blast be interrupted during this stage, lest there should be a chance for air to collect in the downcomer and gas-main, and, mixing with the gas, cause a terrific explosion as soon as the blast goes on again. The writer saw such an explosion take place during a blow-in. A 3-inch pipe had been inserted in the tap-hole to carry off the gas to about the middle of the casting-trough, in which a wood fire was burning; this fire kept the gas burning all the time. When it was time to plug the tap-hole this iron pipe would not come out easily. It soon became twisted, and the blast was thrown off so that it could be removed. When the blast was put on again a terrific explosion took place throughout the furnace and all its connections; all explosion doors were thrown wide open and many were broken. Fortunately no one was hurt, but the accident caused a very undesirable delay for repairs at a critical stage of the blow-in.

The steady increase of CH_4 up to 7.4 per cent., shown in the table, is somewhat remarkable, considering that the amount usually found in the waste gas is about 2.30 per cent. The amount of hydrogen is rather high at first, but diminishes rapidly, and disappears altogether in the third sample (the hydrogen was determined as such, and not as moisture). The small amounts of oxygen shown in the table in the first and seventh analyses were doubtless due to the air taken in the sample-tube.

The column of "Total gases found" represents the sum of the percentages of CO, CO_2, H, and CH_4. The remainder of the waste gas was, doubtless, mainly nitrogen from the blast blown into the furnace, but there were also small amounts of other gases, resulting from the burning and distillation of the wood and coke.

At the right of the diagram are lines and percentages representing the average composition of the waste gas of a furnace running on Bessemer iron, under normal conditions. These figures are averages of actual analyses, in which the ratio $\frac{CO}{CO_2}$ varied from 1.32 to 2.55, and the CO varied from 20 to 29 per cent. Since these results were obtained from the writer's limited observations only, they cannot be taken as standard for Bessemer furnace-gas; but they will serve in this discussion for comparison between the blowing in and the normal work.

At the end of the investigation, the amount of CO was nearly down to the normal percentage, but the CO_2 was far below the normal. The ratio $\frac{CO}{CO_2}$ had not yet started on its final decrease towards the 2 per cent. mark, probably because the stock was still "out of reach," owing to "hitches" in a new filling device.

The amount of "Total gases found" was still on its steady increase towards the normal percentage. The marsh-gas was strangely above its normal mark, while the hydrogen had disappeared altogether.

Some dimensions of the furnace as blown in may be helpful for comparison with other experiments, and are therefore here given, as follows:

Height, 85 feet; diameter of bosh, 20 feet 6 inches; height of bosh above tuyeres, 17 feet; height of tuyere-center, 7 feet 3 inches; diameter of hearth, 13 feet 6 inches; stock-line, 15 feet 0 inch. There were sixteen 7-inch tuyeres.

"C" furnace was not a success on these lines, and has since been relined with only eight tuyeres.

The Kytchtym Medal.

BY DR. PERSIFOR FRAZER, PHILADELPHIA, PA.

(Buffalo Meeting, October, 1898.)

FROM the easternmost point reached by the Ural excursion of the VIIth International Geological Congress (the city of Tschéliabinsk, a little more than 30° of longitude east of St.

Petersburg and about 61° east of Greenwich) the Tschélia-binsk-Ekathérinebourg railway passes through the gold gravels of Miass*, the red and yellow post-tertiary clays covered by tchernozem†, and the granite, miassite‡, etc., characterizing the Ilmen mountains, which extend up north to the iron-mining district of Kytchtym. But besides the Ilmen rocks in this gold, copper, and iron district, another characteristic rock occurs, containing anorthite and corundum, besides chromic iron, and various minerals of interest. (See *Livret Guide*, v., p. 35.)

Among the most appropriate souvenirs of their localities which our hospitable Russian entertainers provided, none were more interesting than the medals cast in our honor by various iron-furnaces as specimens of their material and handiwork. In most cases these objects were as carefully finished as if intended for market; but, on the occasion of a banquet offered by the proprietors of the Kytchtym works to the visiting geologists, a very few of the cast-iron medals distributed to the guests retained the sprue, or attached runner, and were not touched by tool or brush after having been taken from the mould.§ (See Figs. 1 and 2.) Nevertheless, the perfection of detail in the design was such as to excite the admiration of the iron metallurgists among the excursionists, and some of the members of the Institute who viewed one of these medals, exhibited with other Russian objects at the Atlantic City meeting, expressed a desire to know more of the nature of the process of manufacture and of the constitution of the metal.

Mr. Andrew A. Blair, of Booth, Garrett & Blair, of Philadelphia, very kindly analyzed a part of the sprue for this paper, and Prof. Karpinsky was good enough to obtain and forward to me a description of the process, and analyses of the ores and iron from the Kytchtym works.||

---

* A town and river of the Eastern Urals, where gold-gravels are washed.

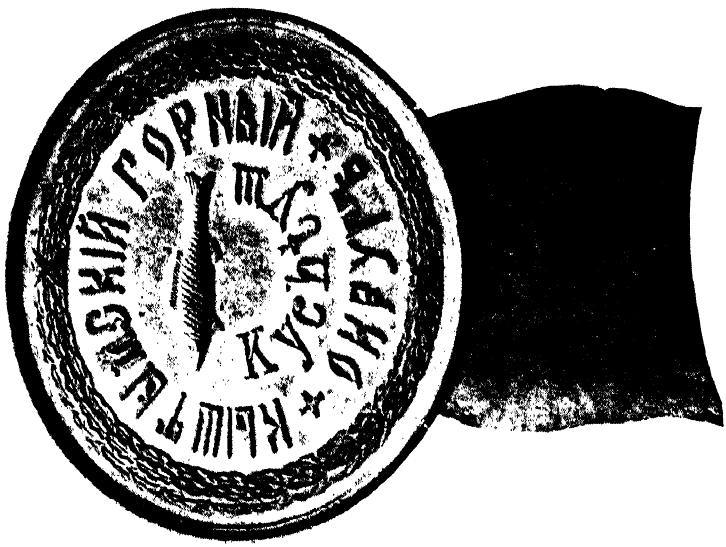
† A dark brown or black soil, formed by the alteration of the superficial rocks.

‡ A biotite syenite, named by G. Rose, containing nephelite or elæolite.

§ The thickest part of the sprue was 2 mm. along its median line, tapering off to 0.3 mm. on the edges. The thickness of the medal was 6 mm. on the edge, and 5 mm. through the smooth disk on which the lettering appears.

|| Not being entirely certain of the exact meaning which Prof. Karpinsky intended to convey in certain technical details, I submitted his letter to an experi-

Fig. 2.



Reverse.

Fig. 1.

Obverse.

The objects known as the Kytchtym iron products are manufactured at the furnace and foundry of Kosliask, a village lying 25 kilometers to the north of Kytchtym. The Kosliask blast-furnace has an open top and is heated with birch-charcoal. The pig-iron from this furnace is remelted in a cupola, provided with an iron chamber above the throat for heating the air, which can thus be conducted into the hearth either hot or cold. The experiment has been tried of using the product taken immediately from the first furnace, but although the iron had quite the same external appearance as that taken from the cupola, it showed a tendency to become white in thin parts, and also to become brittle.

The following are analyses of the ores, limestone and iron, kindly furnished to me by M. Karpinsky:

Analysis of Ores.

	I. Sinarskaia (calcined) Per cent.	II. Pachatnaia. Per cent.	III. Kysyltashskaia. (calcined.) Per cent.	IV. Tchussorskaia. Per cent.
Fe_2O_3	79.86	65.98	66.49	66.32
MnO	0.21	2.19	3.02	1.08
CaO	1.02	1.23	....	0.91
MgO	0.20	0.28	....	0.26
Zn	....	....	....	0.98
Al_2O_3	1.03	2.37	3.45	3.18
SiO_2	6.4	16.76	11.41	16.31
S	0.015	0.02	0.014	0 03
P	0.105	0.297	0.186	0.335
Volatile matter (H_2O)	11.16	10.873	15.43	10.595
	100.00	100.00	100.00	100.00

These ores (brown-hematite) are employed in the following proportions:

enced iron metallurgist, who suggested the modifications which appear in the text. Prof. Karpinsky's letter is in part as follows:

. . . "The Kosliask furnace is heated with birch coal and has an opened throat. The iron with the limestone are introduced into the cupola furnace which has above the throat an air-heating iron vessel, and the air can be conducted into the hearth heated or cold. One has tried to employ the iron taken straight from the furnace, and though the objects made of such an iron had quite the same exterior as those made of iron taken from the cupola furnace, it had sometimes the inclination to whiten in thin parts and grew easy to be broken," etc. . . .

	Per cent.
Sinarskaia,	30.15
Pachatnaia,	38
Kysyltashskaia,	12.57
Tchussorskaia,	12.43
Limestone,	6.85
	100.00

Analysis of the Limestone:

	Per cent.
CaO,	53.78
MgO,	0.61
$Al_2O_3 + Fe_2O_3$,	1.64
SiO_2,	0.94
Volatile matter (CO_2),	42.97
	99.94

Analysis of Iron Taken Directly from the Blast-Furnace:

	Per cent.
Fe,	94.86
Mn,	0.31
C (chemically united),	1.14
Graphite,	2.4
Si,	0.57
S,	0.06
P,	0.48
	99.82

Analysis of Iron Taken from the Cupola-Furnace:

	Per cent
Fe,	93.63
Mn,	0.91
Mg,	traces
C (chemically united),	0.803
Graphite,	3.35
Si,	0.72
S,	0.02
P,	0.447
	99.88

The following is Mr. Blair's analysis of the sprue of the medal itself:

	Per cent.
Iron,	94.063
Manganese,	0.425
Combined carbon,	1.644
Graphitic carbon,	2.530
Phosphorus,	0.642
Silicon,	0.562
	99.866

Mr. Blair adds: "I am sorry the sprue broke as it did; but the metal is exceedingly hard and brittle, as one would naturally expect from the high phosphorus and combined carbon. Although tightly clamped, it broke at the first touch of the saw."

The Relations Between the Chemical Constitution and the Physical Character of Steel.

BY WILLIAM R. WEBSTER, PHILADELPHIA, PA.

(Buffalo Meeting, October, 1898.)

THIS is a subject which our Institute has made peculiarly its own. In the first volume of its *Transactions* the analysis of steel received attention, and every subsequent volume has borne witness to the acuteness and industry of our members in the investigation of this subject. The keynote was struck, in this as in so many other lines of progress, by A. L. Holley, in his paper on "Tests of Steel," read at the Easton meeting of October, 1873 (*Trans.*, ii., 116)—an essay which may rank as a classic by reason of its elegant form, logical force, and clear prevision of all the bearings of its theme. Taken together with the brief report of the discussion which followed it (in which Dr. Drown emphasized the effects of heat-treatment and mechanical handling, and Dr. Sterry Hunt the importance of minute variations in chemical composition, and of possible isomeric and allotropic conditions), it constitutes, after the lapse of twenty-five years, an admirable and adequate introduction to the long list of technical papers which have succeeded it.

Three years later came the epoch-making papers of Dr. C. B. Dudley on the relation between the chemical composition and physical properties of steel rails, which, with the voluminous discussion they evoked, constitute a special volume, published by the Institute, and full of useful suggestions. If Dr. Dudley could be induced to resume this investigation, he would doubtless be able to solve many points still uncertain or disputed.

But I cannot undertake to review the history of this subject, as represented in our *Transactions*—still less to restate all the contributions to it which have been made in books or in papers before other technical societies, since the initiative was taken by the Institute. My more modest purpose is to indi-

cate, by a limited summary, the various steps in the discussion of a particular phase of the inquiry to which, in previous papers, I have devoted some attention. I mean the attempt to find empirical rules by which (due allowance being made for heat-treatment and mechanical manipulation) the tensile strength of steel may be predicted from its chemical analysis.

The provisional establishment of such rules would be useful to producers of steel as furnishing a valuable guide in practice, and to consumers as enabling them to procure intelligent specifications. For the purposes of scientific inquiry, on the other hand, it would be merely a preliminary step in the inductive process of discovering from accumulated observations the ultimate law. That is, it would be simply an arrangement of the vast body of observed facts which would facilitate a further induction from them. I think this point has been misunderstood by some authorities. Nobody pretends, for example, that the effect of each element in steel is exactly represented by an arithmetical addition to, or subtraction from, the tensile strength of the steel. The effect of other elements present, in modifying the effect of the one under consideration, is admitted. Nor is it pretended that, after such modification, a formula expressing the real law could be expressed in terms of simple addition or subtraction.* If mathematically expressible at all, such a comprehensive formula would doubtless involve complicated functions, high powers, etc. Meanwhile, if the ultimate law is ever to be discovered, empirical arrangements and summaries of the facts are the necessary preliminaries to such a discovery, and should not be despised by theoretic investigators. Such empirical rules I have in former papers set forth. Others have suggested other values for the different elements involved. I have no disposition to insist upon mine, which, though based upon numerous careful observations, and proved by many subsequent tests to be approximately reliable in practice, are still open to correction, and will be unhesitatingly withdrawn whenever any others shall be shown to fit observed facts more closely.

I may here remark that the suggestions of other experts in this line, from Dr. Dudley down, have, so far, involved the

* See, in this connection, Dr. R. W. Raymond's remarks, *Trans.*, ix., 605.

same assumption as my own, namely, that of a simple arithmetical form. In order to bring the subject in this confined and limited aspect down to the present time I shall take the liberty of making a few quotations from leading authorities and critics.

In 1885, Mr. P. G. Salom wrote :*

"The most important chemical difference between cast-steel for castings and ordinary open-hearth or Bessemer steel is in the amounts of silicon they contain. Many eminent authorities maintain that silicon is a hardener, and increases, therefore, the tensile strength, like carbon (although in a lesser degree) ; but I have not found this to be the case in my experiments. On the contrary, I have always found it to diminish the tensile strength, and when above 0.5 per cent., to destroy almost entirely the elongation or ductility, making the metal very red-short, and brittle when cold. It may have been that the silicon in the steel that we tested was present as silicic acid, but this could hardly be the case in samples made by the crucible-process in black-lead pots. Such steel made from the best Bessemer muck-bar, to which had been added sufficient ferro-silicon to make over 0.5 per cent. of silicon in the steel, only showed a tensile strength of from 40,000 to 50,000 pounds per square inch in perfectly solid test-bars, whereas the same mixture with less silicon (but higher manganese, however,) invariably gave higher tensile strength. The only explanation that I can suggest which will at all account for the exactly opposite conclusions of the above-mentioned eminent authorities is, that it is probable that silicon exists in steel both as combined and as graphitoid silicon. In the former case it might act like combined carbon and be a hardener ; in the latter it would act like graphite, and undoubtedly would be at least indirectly, or, so to speak, negatively, a softener.

"Another important difference is the comparative wide limits between which the carbon, silicon and manganese may vary in castings without affecting to an important degree the physical results. Such wide variations in steel rails or plates are now quite unknown.

"The influence of carbon on steel is better known than that of any other substance which enters into its composition. No one, however, so far as I am aware, has done anything more than formulate the general law that tensile strength increases with the carbon, other things being equal. I have made the interesting observation that this increase is almost exactly 1000 pounds for every 0.01 per cent. of carbon. That is to say, assuming 0.01 per cent. of carbon to be a unit of carbon, then if to 45,000 pounds (the tensile strength of pure wrought-iron) we add as many thousand pounds as there are units of carbon, we shall be able to make a very close approximation to the tensile strength. Boiler-plate steel, for example, has about 0.15 of carbon, and 15,000 + 45,000 = 60,000 pounds, or about the tensile strength of boiler-plate steel. Rail-steel has about 0.30 carbon and 30,000 + 45,000 = 75,000 pounds, or about the tensile strength of rail-steel. Again, crucible steel contains from 0.50 to 0.85 carbon, from which numbers we get, in the same way, 95,000 and 130,000 pounds, respectively, which include the range of tensile strength of various kinds of tool-steel. Still again, a sample of spring-steel showed 1.0 carbon ; its tensile strength should, therefore, be 145,000. Its actual tensile strength, as tested at Altoona, was 143,000 pounds.

* *Trans.*, xiv., 126.

"Of course this law only holds good where other things are equal. An undue amount of one or all of the other foreign substances that enter into the composition of steel, or unusual physical conditions, would change the results entirely. It may be of value, however, as an indication that, when steel with a known amount of carbon does not possess a certain tensile strength, then the other substances entering into its composition are present in undue proportion, or it must have been made under unusual physical conditions.

"The above law is not applicable to castings, where the presence of so much silicon affects in a notable degree the tensile strength derived from a given amount of carbon, and the physical properties are also affected by the fact that the metal has not been worked."

In another paper* Mr. Salom says that he does not consider phosphorus to be a hardener of steel.

In the latter part of 1885, Mr. Joseph Stokes, Superintendent of the New Jersey Steel and Iron Company, gave me the following easy rule for getting the ultimate strength of steel: "Take 50,000 pounds per square inch as representing the strength of iron, and add to this 1000 pounds for each .01 per cent. of carbon in the steel."

I applied this rule with fair results in 1886–87 to the Clapp-Griffiths steel, manufactured by the Pottsville Iron and Steel Company. I have no doubt that Mr. Salom also had fair results at Chester in using a lower value for iron, and the same addition for carbon, his being open-hearth steel, low in phosphorus; but in each case I think the addition for carbon was too high, and included part of the effect of the manganese present.

Mr. Gatewood, in 1885, in his report on "Mild Steel, used in the Construction of the Hull, Boilers and Machinery of the Dolphin, Atlanta, Boston and Chicago," said:†

"While the possibility of chemical specifications being relied upon for structural steel is very remote, yet, in originating requirements for material, it becomes necessary, especially in connection with intended cost, to appreciate the margin allowed the manufacturer; and although, as we have seen, the physical qualities of steel are affected in many ways, some little understood, by treatment subsequent to tapping, yet, at any particular works, the methods and appliances in use, if not the best, cannot be altered without serious expenditure of time and money, so that for range of physical quality the manufacturer relies solely on the ingredients and proportions of the charge and subsequent treatment in the furnace. With this as the preliminary consideration, an analysis of the results of tensile tests on a chemical basis was undertaken. While, in order to avoid the region of debate, the Board confines itself to the most general discussion of the results, yet

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* *Trans.*, xii., 667. † Pp. 178–194.

it will be readily understood that for even a general analysis of, and useful deduction from, a large number of tests as contained in lengthy tables, an amount of arithmetical labor is necessary such as to deter any but the most persistent investigator from the attempt, so that a summary upon any acceptable basis may well form a valuable addition to the itemized results. . . .

"As a hardener, phosphorus is generally considered to be even more effective than carbon, but its secondary effects are very different. Other things remaining the same, increase of phosphorus, besides raising the tensile strength, notably raises the elastic ratio, diminishes elongation, and more especially diminishes the reduction of area. Its effect in diminishing elongation, and probably also reduction of area, appears to be largely dependent on the amount of other elements present, especially of silicon. For convenience, the latter element must be supposed to vary little, and be present in quantity not above .04 per cent., as is common in open-hearth steels. The effect of phosphorus is then identical in nature to that of cold rolling or finishing, and is to be taken account of in much the same manner. Thus, in estimating the effect of departure from the normal elastic ratio, the influence of variations of phosphorus will have been taken account of. Further, high phosphorus metal is more influenced by cold rolling than steel with less of that element. Once quantified, under the condition of low silicon, the effect of a given absolute amount of phosphorus may probably be relied upon as practically constant.

"Not so with the other most variable element, manganese. The basis from which the effect of variation is to be estimated is itself variable. To illustrate this, it is only necessary to make use of observed variations in the rolling quality of the metal, which is especially influenced by manganese. Thus, of two steels of precisely the same final composition with respect to carbon, phosphorus, manganese, and sulphur, but made from different stock, or in different ways, one may roll well and the other badly. What is sufficient manganese for one is insufficient for the other. This difference arises either from the different conditions in which the manganese may exist in the steel—the most probable cause—or the variable amounts remaining of the oxide of iron whose hot-shortening effect the manganese counteracts. Under average conditions of any given method of manufacture, it is probable that a certain amount of manganese is necessary to prevent hot shortness, which may be called the *saturating* amount of manganese, while only the excess over this amount is free to influence the properties of strength and toughness. In different manufactures, the point of "saturation" of manganese will probably be different, while two heats of the same manufacture may be expected to vary from one another in this respect, depending on the stock and treatment in the furnace. The effect of excess of manganese above this point is certainly to increase the tensile strength, but its influence in other respects is not so decided.

"To quantify the effect of phosphorus or manganese above or below the average amounts, we must have a few tests of steel varying only in the amount of one of these elements from the average, from which, by comparison with the carbon curve, a derived curve can be constructed for the effect of such variations.

"The same method must be used to quantify the effect of departure from the normal elastic ratio previously corrected for variation of phosphorus. . . .

"This steel shows low manganese and low elastic ratio, the latter probably corresponding to comparatively rapid and hot finishing. Accurate carbon determination of the test-piece gave .10 per cent., for which the normal elastic ratio is 65.53 per cent. Estimating the effect of defect of elastic ratio below normal value as somewhat less than that of excess, the difference of tensile strength correspond-

ing to the defect of elastic ratio of 5.58 per cent. may be stated at about 3000 pounds; or, under average conditions of finishing, the tensile strength would be about 58,400 pounds. As the tensile strength under these conditions, with average manganese, would be 59,270 pounds, the defect of .24 per cent. of manganese below the average of .45 per cent. diminished the tensile strength by not more than 870 pounds, provided none of the defect of elastic ratio is due to the low manganese. An equal excess of manganese above the average value has a much greater influence on the tensile strength, the *saturating* amount for this steel being believed to be near .40 per cent. . . .

"Combination of comparison of the curves obtained for the three steels is not possible with any accuracy, the conditions of physical treatment, manufacture and carbon determinations being too widely different. It appears probable, however, that open-hearth mild steel of about .07 per cent. phosphorus, tested under the average conditions of this inspection, should show an average increase of tensile strength of not far from 650 pounds per square inch for an increase of .01 per cent. of carbon in the finished product with a simultaneous decrease of from .10 to .12 per cent. of final elongation in a piece of average dimensions, 60,000 pounds tensile strength and 27 per cent. elongation corresponding to about .10 per cent. of carbon by color-test in the test-piece itself. Large variations from these average results are to be expected from varying conditions of treatment and manufacture."

Mattieu Williams* says:

"My own experiments on what may be termed 'pseudo-steels,' *i.e.*, iron combined with something else as a substitute for the carbon of true steel—indicate that phosphorus has about three times the hardening effect of carbon; thus, a sample of pseudo-steel containing 0.3 per cent. of phosphorus is about as hard as true steel containing 0.9 of carbon; that silicon has a hardening effect about intermediate, *i.e.*, 0.3 of silicon is about equivalent to 0.45 of carbon. I say 'about' because such estimates can only be approximate, partly on account of the difficulty of obtaining such compounds free from other impurities, and partly from the roughness of the available tests for hardness.

"References to chemical text-books and technological treatises of different dates supply a number of curious and instructive examples of the prevalence of the fallacy of estimating the value of steel by its hardness.

"A little reflection will show that the ordinary duties of steel demand a special ability to endure smart blows or vibratory shocks; the teeth of a saw or file, the edges of chisels, planing-irons, knives, and other edge tools, are continually receiving such shocks, and are ripped or notched unless they combine their hardness with remarkable toughness.

"The same kind of endurance is demanded of steel used in construction. Why did the Tay Bridge give way just as the train came upon it? Was it merely the additional weight of the train? or was it chiefly due to the vibration produced by the train acting at the moment of extreme tensile strain produced by the wind on that fatal, stormy night?

"In spite of all that has been said by high authorities concerning the improvement of steel by the addition of silver, platinum, chromium, palladium, titanium, tungsten, rhodium, silicon, manganese, etc., I maintain that for general tool-making, and for every purpose where tenacity has to be combined with stability,

* *The Chemistry of Iron- and Steel-Making and their Practical Uses*, London, 1890, p. 175.

and hardness with toughness, *the best material at present known is made by combining the purest of iron with the purest of carbon, and excluding everything else.*"

Prof. H. M. Howe,* after giving the results of different investigators, and some of his own, states:

"While we cannot accurately quantify the effects of carbon, I believe that for ordinary unhardened merchantable steel, the tensile strength is likely to lie between the following pretty wide limits:

Carbon. Per cent.	Ultimate Strength. Lbs. per sq. In.	Range of Variation.† Lbs. per sq. in.
.05,	50,000 to 66,000	16,000
.10,	50,000 to 70,000	20,000
.15,	55,000 to 75,000	20,000
.20,	60,000 to 80,000	20,000
.30,	65,000 to 90,000	25,000
.40,	70,000 to 100,000	30,000
.50,	75,000 to 110,000	35,000
.60,	80,000 to 120,000	40,000
.80,	90,000 to 150,000	60,000
1.00,	90,000 to 170,000	80,000
1.30,	90,000 to 115,000	25,000"

He further sums up the whole matter as follows: ‡

"In the present state of our knowledge it seems probable that the conditions in a solidifying steel ingot, and perhaps in many other alloys and similar compounds, resemble those in a solidifying crystalline rock. For we find that the chemical condition of the components of the solidified steel and the size and probably the shape and arrangement of its individual crystals are affected according to unknown laws by changes in its ultimate composition, and by the conditions which precede and accompany its solidification and cooling. . . .

"The influence of the conditions of cooling on the structure of steel is readily recognized. Slow, undisturbed cooling induces coarse crystallization; if the metal be vigorously hammered during slow cooling, the structure becomes much finer; if the cooling be sudden, extremely fine structure results. That other and now unguessed conditions profoundly alter both the mineral species and the structure of steel, and that changes in ultimate composition modify both species and structure of steel, as of crystalline rock, in most complex ways, is indicated by the utterly anomalous relations between the ultimate composition and the mechanical properties of steel. This anomalousness, which has puzzled so many, is readily explained by the close resemblance between the conditions of the formation of rock and of ingot, which not only shows us why we do not discover these relations, but that in all probability *we never can* from ultimate composition. The lithologist who attempted to-day to deduce the mechanical properties of a gran-

* *The Metallurgy of Steel*, New York, 1890, p. 16.

† This column I have added. It shows that Prof. Howe regards the range of variation as different for different percentages of carbon, being greatest at 1 per cent.

‡ *The Metallurgy of Steel*, pp. 3 and 4.

ite from its ultimate composition would be laughed at. Are our metallurgical chemists in a much more reasonable position?

"The complex way in which slight changes in ultimate composition may induce disproportionate changes in the proximate composition of the mineral species making up the solid steel, and through them its mechanical properties, is readily seen on reflection. If between the elements of the molten mass there exists a certain balance which just permits the formation of certain compounds during solidification, the introduction of a minute quantity of a certain element, say manganese, might just upset this balance and give rise to the formation of quite a different set of compounds, which might have radically different effects on the properties of the metal. While, were the original composition somewhat and perhaps but slightly different, then the addition of the same quantity of manganese might not in the least alter the kind or proportion of the different mineral species which make up the solid mass.

"If, pointing out that .02 per cent. phosphorus sensibly alters the ductility of steel, you ask how this effect can be due to so minute a quantity of a simply intermingled mineral, I answer: (1) That we have just seen how minute changes in ultimate composition may profoundly alter the proximate composition. One per cent. of salt distributed through gneiss would destroy its weather-resisting powers; 5 per cent. of mica would give it strong cleavage; so 5, or even 1 per cent. of a mineral whose presence in steel might be due to an addition of say .02 per cent. of phosphorus, might profoundly alter its properties. We note among the hydro-carbons compounds whose physical properties differ greatly, yet whose ultimate composition is very similar, nay even identical. (2) That if 0.0002 per cent. of iodine gives starch liquor a perceptible color it is not surprising that 100 times as large a quantity of phosphorus should perceptibly affect the properties of the iron matrix with which we may fancy that it directly combines. (3) That even so minute a quantity of phosphorus as .02 per cent. may so affect the conditions of solidification, for example by altering the fluidity of the matrix at some critical temperature at which crystallization occurs, as to greatly affect the size, shape and mode of arrangement of the crystals of some of the minerals present, and of the matrix itself

"If now it is asked why, if these so-called minerals form in steel during solidification, we never see them, I reply: (1) That the component minerals of many crystalline rocks are only discernible under the microscope, and even then only because they happen to be more or less transparent, to differ from each other in color, and to have crystalline forms which have been accurately determined by the study of large crystals; (2) That we have hardly begun to look for them in steel; (3) That under favorable circumstances we do find what appear to be distinct minerals in steel (graphite, Fe_3C, TiC in definite crystals), and to so great an extent as to render it probable that these or similar minerals usually exist, but that, being opaque, so nearly alike in color, and in such minute and uniformly distributed particles, they escape observation. In considering segregation, we shall see that when steel contains considerable quantities of manganese, phosphorus, sulphur, etc., what are probably distinct minerals, perhaps even of definite chemical composition, form, now concentrating in the centre of the ingot, now liquating from its exterior according to the existing conditions.

"If these views be correct, then, no matter how accurate and extended our knowledge of ultimate composition, and how vast the statistics on which our inferences are based, if we attempt to predict mechanical properties from them accurately, we become metallurgical Wiggenses. For while we may predict that siliceous rocks will usually be vitreous, July hot, April rainy, and phosphoric

steel brittle, yet when we go farther and predict accurately, we state what is not inferable from our premises. It may, and sometimes does, snow in July; Christmas may be warmer than Easter; the more siliceous may be less vitreous than the less siliceous rock, and the more phosphoric steel tougher than the less phosphoric one.

"And here it may be observed that the intimate knowledge which the public and many non-metallurgical engineers attribute to metallurgists, as to the effects of composition on physical properties, has, I believe, no existence in fact. Many steel-metallurgists persuade themselves, from wholly insufficient data, that they have discovered the specific quantitative effects of this or that element; in other, and I trust fewer, cases, in metallurgy as in medicine, the charlatan feigns profound knowledge, dreading the effect on his client of acknowledged, though unavoidable, ignorance. Many an experienced steel-maker has confidently assured me of such and such specific effects, producing, when challenged, a few analyses unconsciously culled from those which opposed his view, and shown, on comparison with a larger number, to be without special significance.

"When we confront him with cases which upset his theory, he calmly replies that if we had only determined the sulphur as well, all would have been clear. If, by bad luck, this, too, is known, he thinks, probably, that nitrogen or carbonic oxide may affect matters; or, possibly, he attaches great weight to oxygen, which he can always fall back on, triumphantly remarking that when we can determine this element the problem will be solved. . . .

"By what methods ultimate composition is to be determined is for the chemist rather than the metallurgist to discover. But, if we may take a leaf from lithology, if we can sufficiently comminute our metal (ay, there's the rub!), by observing differences in specific gravity (as in ore-dressing), in rate of solubility under rigidly-fixed conditions, in degree of attraction by the magnet, in cleavage, luster, and crystalline form under the microscope, in readiness of oxidation by mixtures of gases in rigidly-fixed proportions and at fixed temperatures, we may learn much.

"Will the game be worth the candle? Given the proximate composition, will not the mechanical properties of the metal be so greatly influenced by slight and undeterminable changes in the crystalline form, size and arrangement of the component minerals so dependent on trifling variations in manufacture as to be still only roughly deducible?"

There is certainly very little here to encourage one in continuing the investigation. But since 1890, when Mr. Howe wrote the above, there have been great advances made in the methods of investigation, and our actual knowledge of steel has been increased. We also understand better the effects of the heat-treatment of steel, and he will, I think, agree that the case is now not as hopeless as it formerly appeared.

The following extract is made from the discussion of Report to the Alloys Research Committee of the Institution of Mechanical Engineers, London.*

* *Proc. Inst. of Mech. Engineers*, 1891, p. 588.

"Mr. Charles E. Stromeyer had recently compiled the accompanying table, which might be of interest in connection with this discussion, for showing generally the influences of the various impurities in mild steel. These data he had collected from papers read before various societies; but with many of them he

Influence of Impurities on Some of the Properties of Mild Steel.

Element Constituting Impurity.	Tenacity.	Elongation.	Resistance to Impact.	Pliability. Cold.	Pliability. Red Hot.	Pliability. White Hot.	Welding.	Hardening.	Corroding.
Carbon........	+	–	–	–		–	–	+	–
Silicon........	+	–	–		–	–?	0	+	
Arsenic........	+	–	–	–	.	0		–	
Phosphorus..	+	–	–	–	0	–			
Sulphur......	0	0		0	–	0	–		
Copper........	+	–			–	–	0?		
Manganese...	+	+	+		+	+	0	–	?
Nickel........	–	+	+	+				–	–
Chromium...	+	0	+	+				+	

+ denotes that the property is increased by the impurity.
– " " " diminished by the impurity.
0 " " " not affected " "
? " " various authorities are conflicting.

did not agree, particularly in those cases where the authorities were conflicting. In this compact form, however, he hoped the table might prove useful, and might be completed and corrected. Amongst the doubtful cases was silicon, of which it had not been definitely ascertained whether it diminished the pliability of mild steel at a white heat, or, in other words, whether it produced red shortness. Similarly, the evidence was conflicting as to whether copper affected the welding qualities of steel, and also as to whether manganese affected its susceptibility to corrosion. The table might also with advantage be extended to include not only other elements, but also other properties, such as melting-point, viscosity, limit of elasticity, toughness, etc. Moreover, it appeared to him that no one impurity could be said to produce a certain effect when other impurities were present. It seemed as if some of the ingredients intensified, reduced or entirely reversed the effect of others. Thus, the tenacity and hardening qualities were increased, and the elongation and pliability reduced, by carbon, while a further admixture of manganese reduced or reversed all these effects except the increased tenacity. Tungsten and some other ingredients not included in the table appeared to agree with silicon and chromium in increasing the property of hardening which carbon gave to iron, and to reduce some of the other properties."

Mr. Vosmaer says:*

* *The Mechanical and Other Properties of Iron and Steel in Connection with their Chemical Composition*, London, 1891, p. 17.

"The tensile strength of steel is directly variable with its carbon-content. If the breaking-loads are set out as ordinates and the contents of carbon as abscissæ, then the line uniting corresponding points is nearly a straight one. For pure carbon-steels—that is, steels in which carbon forms the chief constituent, and in which only minor quantities of other elements are present—it may be said that each 0.1 per cent. carbon, more or less, increases or decreases the ultimate strength by about 6 kg. per sq. mm. A maximum of strength is obtained at about 1 per cent."

Therefore each 0.1 per cent. carbon equals 835 pounds per square inch.

"The effect of 1 per cent. manganese on the tensile strength is nearly equal to 0.2 per cent. carbon, thus being about one-fifth of it. When the carbon is kept constant, and the percentages of manganese set out as abscissæ and the corresponding breaking-loads as ordinates, the union of the corresponding points is nearly a straight line, the maximum being at about 3 per cent. manganese. On annealed steel manganese may be considered to be about equal to one-third carbon; thus, each 0.1 per cent. manganese, more or less, will cause a rise or fall in the ultimate strength of 1.8 to 2 kg. per sq. mm.

"More rapidly than the ultimate strength is the elastic limit raised by increasing the manganese; thus manganese, when in larger quantities (more than 1 per cent.), causes brittleness. The elastic limit lies at 50 to 55 per cent. of the ultimate strength when not exceeding 0.6 per cent. manganese.

"As regards ductility, the influence of manganese is only one-seventh to one-eighth of carbon; thus, of two steels having the same ultimate strength, the manganese steel will have larger elongation than the carbon steel, provided the manganese is not more than about 0.6 or 0.8 per cent. Whilst 0.1 per cent. carbon more causes a loss in elongation of about 4 per cent., 0.1 per cent. manganese more causes a loss of about 0.5 per cent. only; hence it may be advantageously used in structural steel, where both strength and ductility are required.

"It is easier to obtain certain strength- and elongation-figures when replacing part of the carbon by manganese than by exclusively using carbon—the severer the reception-test, the easier fulfilled when using manganese—but the limit is soon reached because of the rise of the elastic limit."*

That is to say, each 0.1 per cent. of manganese represents 170 pounds per square inch.

"Poor silicon has been scolded for a long time, but later investigations have shown this to be quite wrong, and that silicon is not so bad, after all. Silicon increases tensile strength, however, in a much less degree than carbon, being only about one-tenth of this—0.1 per cent. silicon more giving a rise in the ultimate strength of 0.5–0.6 kg., whereas 0.1 per cent. carbon gives 5–6 kg. increase."†

Therefore each .01 per cent. silicon equals 85 pounds per square inch.

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* *The Mechanical and Other Properties of Iron and Steel in Connection with their Chemical Composition*, 1891, p. 43.

† *Ibid.*, p. 75.

"We have seen already how the influence of phosphorus rapidly increases with the carbon. This may be illustrated by another example. As already said, tool-steel seldom contains more than 0.02 per cent. phosphorus, whereas rail-steel may have 0.1 per cent., and under certain conditions even as much as 0.4. It is true both steels have quite different duties to perform, but undoubtedly 0.1 per cent. phosphorus in a hard steel would be fatal for whatever purpose it might be used. Phosphorus increases the ultimate strength to a degree about one-fourth of carbon. Thus, 0.1 per cent. phosphorus, more or less, causes a rise or fall of 1.5 kg. in the breaking-load; a maximum lies at about 0.3–0.4 per cent., but the elastic limit rising more quickly, any increase in phosphorus will diminish the ductility, hence phosphorus makes steel brittle."*

That is to say, each .01 per cent. of phosphorus equals 213 pounds per square inch.

In a former description of my work of 1892†, I said:

"Before attempting to investigate the effects of carbon, phosphorus, etc., on the ultimate strength of the steel, I had to find out how the ultimate strength was affected by the finishing temperature in rolling. This varies with thickness and width of plates, even when great care is taken to control the same. By rolling parts of the same heat into plates of different thicknesses the following values were arrived at. Assuming a $\frac{3}{8}$-inch plate under 70 inches wide to give normal results, we have corrections for size of plates shown in the following table:

Corrections for Size of Plates.

Thickness of Plates. Inches.	Up to 70 in. wide. Tensile Strength. Lbs. per sq. in.	Over 70 in. wide. Tensile Strength. Lbs. per sq. in.
$\frac{3}{4}$,	—2000	—1000
$\frac{11}{16}$,	—1750	— 750
$\frac{5}{8}$,	—1500	— 500
$\frac{9}{16}$,	—1250	— 250
$\frac{1}{2}$,	—1000	— 0
$\frac{7}{16}$,	— 500	+ 500
$\frac{3}{8}$,	— 0	+1000
$\frac{5}{16}$,	+3000	+4000

"On this point my results are not very satisfactory, but during the investigation the importance of controlling the temperature at which the plates were finished was brought out forcibly."

And, in another part of the same paper‡, I observed:

"When rolling heavy steel plates, trouble is often caused by finishing them at too high a temperature, which gives a material with crystalline fracture, poor reduction of area, and poor bends. In order to guard against this and control the

* *The Mechanical and Other Properties of Iron and Steel in Connection with their Chemical Composition*, 1891, p. 96.

† *Jour. Iron and Steel Inst.*, Part i., 1894, p. 328. ‡ *Idid.*, p. 335.

finishing temperature we use very light draughts in rolling, and produce as good results in heavy plates as in the light ones. Too much importance cannot be given to the heat treatment of steel. Prof. H. M. Howe's experiments on the subject are of the greatest value, and it is to be hoped that they will be continued on a larger scale, in connection with the work of rolling and forging."

On starting my investigation at Pottstown I gave carbon a value of 1000 pounds and then decreased it to 900, 800, etc., but found that I had to take phosphorus into account; and I worked for some time with carbon and phosphorus, giving each the same value and different values, and with different bases for the strength of pure iron, from 40,000 pounds upward, but could not get results that were satisfactory to me. I then gave a value to manganese, and soon found much improvement in my work. In order to investigate the matter more thoroughly, I recorded each test on a separate card, so as to facilitate grouping the tests under any given element. For instance, taking .15 per cent., and placing all the tests of that carbon in one pile, I eliminated the effect of carbon in that particular lot of tests, as far as the differences were concerned between the tests; and, by giving each of the other elements a value, I tried to account for these differences, and proceeded in the same manner in grouping the cards under other elements. Thus the values were arrived at, which I gave in my first paper, in 1892.* But after this, as the number of tests increased, I found that by giving sulphur a value the results were improved; and my tables allowing for sulphur were given in a paper before the International Engineering Congress, at Chicago, in August, 1893.† I found, early in the investigation, the importance of making a correction for the thickness of the plates, and my original 480 tests, with the estimated ultimate strength corrected for size, are given in detail in the latter paper, in which I said:

"In basic open-hearth steel, we have deducted 2100 pounds from the estimated ultimate strength; this has given fair results, but the amount of deduction may have to be modified in using the new table.

"From the results obtained, I believe that I am safe in saying that in all rolled steel the quality depends on the size of the bloom, or ingot, from which it is rolled, the work put on it, the temperature at which it is finished, and the chem-

* *Trans.*, xxi., 766.

† *Ibid.*, xxiii., 113.

ical composition of the steel; that is, a table of this kind could be used for beams, angles, bars, etc. For instance, a 6 by 6 by $\frac{3}{8}$-inch angle, with a given chemical composition, might give 4000 pounds higher ultimate strength than indicated by my table; but by making this allowance, the table could be used to advantage to show what ultimate strength another heat of steel with different chemical composition would give if rolled into the same sized angle. I trust that this point is clear, and that some of the shape mills will take the matter up and let us hear from them."

I now give the same tables in a more detailed form, in order that they may be more thoroughly understood, and at the same time may be in better shape for use in comparing my results with the work of other investigators.

The values for carbon are given up to .60 per cent., and the values of manganese up to 1 per cent. These values may not apply to the high steels; but, having them in convenient form to use, we can, by trial, soon find out what corrections and modifications are required, bearing in mind, of course, that the difference in finishing-temperature has more effect on the high-carbon steel than on the ordinary steels of 70,000 pounds tensile strength and under. This should be recognized in drawing up specifications for the high steels; and the limits between the high and low ultimate strengths specified should be greater than the ordinary allowance of 8000 pounds specified for the steel of 70,000 pounds and under.

In Table I. I give in the first column the values from .05 to .60 per cent. carbon at 800 pounds for each .01 per cent. of carbon, and in the second column the corresponding values for this carbon, increased by the strength of pure iron of 38,000 pounds, as used in my first paper. In this case sulphur was not considered. In the third column are given the corresponding values for the same carbons, increased by 34,750 pounds for pure iron, as used in my second paper, where the value of sulphur was considered.

In Tables II. to IX., inclusive, I give the increase in ultimate strength for each .001 per cent. of phosphorus from 0 to .109 in each case. The increase due to .01 per cent. of phosphorus in the presence of .06, .07 or .08 per cent. of carbon is 800 pounds, as shown in Table II. It increases from this up to 1500 pounds for each .01 per cent. of phosphorus in the presence of 0.15 per cent. of carbon, or over, as is shown in Tables III. to IX., inclusive. Each table has sufficient explana-

tion in heading to make this matter plain and show how it is constructed. In all cases take the hundredths of per cent. in the first column to the left, and follow along this horizontal line until column is reached having the thousandths of per cent. sought. This will give the increase in tensile strength due to phosphorus alone.

In Table X. I have given a summary of the value of phosphorus in the first column of Tables II. to IX., inclusive, and at the foot of each column is given the increase due to each .001 per cent. of phosphorus. In each case this increases from 80 pounds in the presence of .06, .07 and .08 per cent. up to 150 pounds in the presence of .15 per cent. of carbon.

Table XI. is similar in every respect to Table X., except that the value of base of 38,000 pounds for pure iron, and an increase of 800 pounds for .01 carbon, have been added to it in every case. This is the table that was given in my first paper. The increase due to sulphur is not considered in connection with this table. At least, no allowance was then made for high or low sulphur.

Table XII. is the same in every respect as Table XI., except that the value of the base of 34,750 pounds for pure iron has been used in every case. This is the table given in my second paper, where the increase of 500 pounds for each .01 per cent. of sulphur is taken into account. It is to be used in connection with Table XIV. for sulphur.

In Table XIII. I have given the increase in ultimate strength for each .005 per cent. of manganese from .15 up to 1 per cent. of manganese. In my former papers I have explained that the results were much better when the effect of manganese was not considered a constant per unit from .15 per cent. upward, but that the first increments seemed to have more effect in increasing the ultimate strength than the same number of units have as the manganese increases. Except that it has been enlarged up to 1 per cent., this is the same as that given in my previous papers.

In Table XIV. I have given the increase in ultimate strength for each .001 per cent. of sulphur from 0 to .109 per cent. based on an increase of 500 pounds for each .01 per cent. of sulphur. This table is the same as given in my previous papers, and is to be used in connection with Table XII.

Mr. F. Osmond says:*

"It may be fairly said that a steel derives all its mechanical properties from its chemical composition, its molecular condition and its structure. This statement is not exactly novel; it was put forth about 1878 by the engineers of the Terre Noire works, who, in their turn, had predecessors holding the same view. Yet the proposition has remained in dispute, and has received but scanty recognition. Thus it has been sought to determine the relations between the variables; for instance, between the chemical composition and the wear of rails, or even between the amount of rolling and the resistance to shock; relations which unquestionably exist, but which are most frequently masked in industrial products by irregularities of structure. Now that the essential influence of structure has been made plain for all time to come, we may hope that this new independent variable will be introduced into such investigations, or eliminated beforehand, so that, to the greatest benefit of both producers and consumers, the relations may be ascertained which connect the qualities of steel with their chemical, physical and mechanical co-efficients, as determined by the tests of manufacturers and inspectors."

Mr. Albert Sauveur, in 1893,† laid down an interesting and important series of propositions concerning the effect of heat-treatment on the molecular structure and composition of steel.

While in Chicago, in that year, I saw the importance of having some definite plan for our discussions on the physics of steel. In order to avoid the necessity of going into many details, and explaining from time to time that we understood the bearings of many points on the matter, I took this subject up with Mr. Sauveur and Prof. Howe, who were much interested and saw the importance of it. Mr. Sauveur took my rough plan and enlarged it, and his suggested lines for discussion were submitted to Prof. Howe, who framed the final schedule, as given.

I refer to this matter at the present time because Mr. William Metcalf, in his discussion of Mr. Cunningham's recent paper before the American Society of Civil Engineers, treats the subject as though we had all overlooked the changes in the physical tests of material produced by heat-treatment. This is rather discouraging, after the consideration that has been given to these very changes by most of us, and the plea made from time to time for more experiments on *the heat-treatment of steel in connection with work.* Mr. Metcalf's remarks were as follows:‡

* *Trans.*, xxii., 260.

† *Ibid.*, xxii., 546.

‡ *Trans. Am. Soc. C. E.*, xxxviii., 84.

"There has been for many years a very strong belief that there is some relation between the strength of any piece of iron and its composition, which would be a guide for estimating the strength of the metal from its chemical composition. Engineers should be very grateful to Messrs. Webster and Campbell for the extraordinary amount of work they have done in trying to work out such an exceedingly complicated and difficult problem. Their work is admirable, and may be very useful if it is not applied practically until it has been carried much farther. If it is considered that the manufacturer can vary the strength of steel, by manipulation, from 50 per cent. to 100 per cent. one way or the other, it is evident that the formulas are not safe guides for determining the strength of steel of a given composition. It will not do to lay aside the actual tests of material. Another difficulty in the way of the use of the formulas is the fact that they apply only to the grade of steel with which the experiments were conducted. Mr. Campbell states that this was the mild structural steel that he was making."

To which my answer was:*

"In the discussion Mr. Metcalf remarks that the manufacturer can vary the strength of steel, by manipulation, from 50 per cent. to 100 per cent. one way or the other. I do not consider the criticism a fair one, as it is the manufacturer of structural steel who depends, more than anyone else to-day, on the relation between the chemistry and the physical properties of steel. They all apply the steel to the orders by its chemical composition, and if the tension-tests of the finished product do not give the results that they expected from the analyses of the heat of steel, they at once examine into the conditions of heating and rolling; and it is by this close watch of both the chemical composition and physical treatment of the steel that they have made such great advances in its manufacture during recent years. Mr. Metcalf fails to state that in producing the changes referred to above, the manufacturer, in so doing, would also greatly change the elastic limit, the percentage of stretch, the percentage of reduction of area, and the bending properties of the steel treated. These changes for any given treatment would be greater or less in accordance with the chemical composition of the steel so treated, and would emphasize the effects of the carbon, phosphorus, manganese, etc., on the steel. Mr. Metcalf would not consider it unreasonable to state that, if a piece of steel with a known chemical composition and a known treatment in heating and rolling produced, say, 65,000 pounds tensile strength, at a future time another heat of steel with exactly the same chemical composition, with the same treatment in heating and rolling, will produce about the same results—namely, 65,000 pounds ultimate strength. If we had enough standard tests to cover all cases of structural material, and could properly record them, it would be merely a matter of turning to such records in order to know what a given heat of steel should produce in the finished product, without having to interpolate for the cases that we have not so recorded. It is this interpolation that we have been trying to arrive at by giving values to each of the elements. The problem is certainly a very difficult one, and we are still groping somewhat in the dark; but we must agree that marked progress has been made, when we consider that only in 1892 no less an authority than Prof. H M. Howe, in summing up this whole matter, said: 'If these views are correct, then, no matter how great and extended our knowledge of ultimate composition, and how vast the statistics on which our inferences are based, if we attempt to predict mechanical properties from them accurately, we become metallurgical Wiggenses.'"

* *Trans. Am. Soc. C. E.*, xl., 455.

Mr. Metcalf himself* has recognized, and, indeed, proved, with unsurpassed clearness, both in theory and in practice, the importance of chemical composition, particularly as regards carbon.

In 1895 Mr. H. H. Campbell made an investigation on 1920 pieces of 2 by ⅝-inch bars rolled from small ingots of acid and basic open-hearth steel.† These pieces were tested as they came from the rolls, and drillings were taken from the test-pieces. The results were divided into 137 groups. He says:‡

"After forming these groups, the average manganese, sulphur, phosphorus and ultimate strength of each were calculated from the records, while the average carbon, silicon and copper were determined by weighing an equal quantity of drillings from each bar, and making a chemical analysis, the carbon being determined by combustion."

Mr. Campbell attempted by the method of least squares to work out the values for each element, but his results were unintelligible, and he remarks:§

"It is with no little disappointment that I am forced to confess that further investigation throws grave doubts on the validity of this method of least squares when applied to such a number of unknown quantities, and when any one of these quantities is of very little importance."

He then neglected the effect of silicon, sulphur and copper, and obtained several results for each of the elements, in the order in which they are given below, for each class of steel. In the latter part of his investigation he considered a new series of acid and basic steels. This gave him for the acid steel 56 groups in the new series, as against 70 groups in the first series. For the basic steel he had 74 groups in the first series and 72 groups in the new series.

The values he has adopted may be summarized as follows:

Mr. Campbell's Values for Pure Iron and Increase Due to .01 Per Cent. of Carbon, Phosphorus and Manganese, in Pounds per Square Inch.

In b and e for Acid Steels, and in g and j for Basic Steels, the increase due to manganese was not considered. Mr. Camp-

* *Steel: A Manual for Steel-Users*, 1896, pp. 46–51.

† *The Manufacture and Properties of Structural Steel*, New York, 1896, p. 287.

‡ *Ibid.*, p. 288. § *Ibid.*, p. 297.

bell remarks, regarding the value of 1444 pounds for phosphorus in j, "This value of phosphorus was not sustained by any other evidence."

"The factor R represents an allowance for the conditions under which the piece is rolled, whether finished hot or cold. In the present series of groups it is zero."

ACID STEELS.

	C.		P.		Mn.		Pure Iron.		
a Old series, . .	1529	+	1316	+	39	+	34,326		= Ultimate Strength.
b Old series, . .	1485	+	1260			+	33,000		= Ultimate Strength.
c New series, . .	1126	+	716	+	3	+	40,439		= Ultimate Strength.
d Old series, . .	1368	+	1068	—	23	+	37,544		= Ultimate Strength.
e Both series, . .	1210	+	890			+	38,600	+ R	= Ultimate Strength.

BASIC STEELS.

	C.		P.		Mn.		Pure Iron.		
f Old series, . .	1035	+	941	+	53	+	38,996		= Ultimate Strength.
g Old series, . .	1085	+	1200			+	40,000		= Ultimate Strength.
h New series, . .	935	+	939	+	114	+	36,335		= Ultimate Strength.
i Old series, . .	1035	+	941	+	53	+	38,996		= Ultimate Strength.
j Both series, . .	998	+	1444			+	39,987		= Ultimate Strength.
k Both series, . .	950	+	1050	+	85	+	37,430	+ R	= Ultimate Strength.

Mr. Campbell estimated the ultimate strengths of each of his groups by several of the above values, and the difference between the estimated ultimate strengths and the average actual ultimate strengths of the groups were small in most cases for each class of steel, but the best results were obtained by the last values given in the above tables.

Mr. Campbell's conclusions were as follows:*

"1. The strength of pure iron, as far as it can be determined from the strength of steel, is about 38,000 or 39,000 pounds per square inch.

"2. An increase of .01 per cent. of carbon raises the tensile strength of acid steel about 1210 pounds per square inch, and of basic steel about 950 pounds. This difference between the effect of carbon upon acid and basic steels, as found by mathematical analysis, is corroborated by the graphic records in Figs. XX. and XXI.

"3. An increase of .01 per cent. of manganese has very little effect upon acid steel unless the content exceeds .60 per cent., but it raises the strength of basic steel about 85 pounds per square inch.

"4. An increase of .01 per cent. of phosphorus raises the tensile strength of acid steel about 890 pounds per square inch, and of basic steel about 1050 pounds.

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* *The Manufacture and Properties of Structural Steel,* New York, 1896, pp. 329–331.

"5. The following formulæ will give the ultimate strength of ordinary open-hearth steel in pounds per square inch, the carbon, manganese and phosphorus being expressed in units of .001 per cent., and a value being assigned to R in accordance with the conditions of rolling and the thickness of the piece:

FORMULA FOR ACID STEEL.

38,600 + 121 Carbon + 89 Phosphorus + R = Ultimate Strength.

FORMULA FOR BASIC STEEL.

37,430 + 95 Carbon + 8.5 Manganese + 105 Phosphorus + R = Ultimate Strength.

"6. The metals, from which these data were derived were ordinary structural steels ranging from .02 to .35 per cent. of carbon, and it is not expected that the formulæ are applicable to higher steels or to special alloys.

"7. A considerable difference may be found between steels which apparently are of the same composition, and which, as far as known, have been made under the same conditions.

"8. In the case of acid steel, an increase in manganese above .60 per cent. will raise the tensile strength above the amount indicated by the formula, the increment being quite marked when a content of .80 per cent. is exceeded.

"9. In steels containing from .30 to .50 per cent of carbon, the value of the metalloids is fully as great as with lower steels, while the presence of silicon in such metal in proportions greater than .15 per cent. seems to enhance the strengthening effect of carbon.

"10. In steels containing less than .25 per cent. of carbon the effect of small proportions of silicon upon the ultimate strength is inappreciable.

"11. Sulphur, in ordinary proportions, exerts no appreciable influence upon the tensile strength.

"12. Both acid and basic steels containing less than .30 per cent. of manganese give an actual strength greater than is shown by the formula, and when this is taken in connection with the abnormal strength of the unusually pure metal shown in Group 198 of Table 131, it is indicated that oxide of iron raises the ultimate strength."

In an investigation of this kind the tests should be considered individually, as, when they are grouped, you get average chemical compositions and a more constant relation between the elements—say, for instance, between carbon and manganese. In this way you are liable to give to one element part or all of the effect of another element. If the elements increase or decrease in the same proportion, the estimated values meet the conditions very well. I tried grouping the tests, and soon found that it was not as satisfactory a way as using the individual tests. I trust that Mr. Campbell will put all of his individual tests on record, in order that others can work on them, and compare his values when applied to each of his tests.

In order to compare Mr. Campbell's values with mine, and

put them in a convenient form to use, I have prepared Tables XV. to XX., inclusive, embodying his values in the same general form as those used in my previous work.

EFFECT OF THICKNESS ON THE PHYSICAL PROPERTIES OF PLATES.

Mr. Campbell says:*

"The effects caused by variations in rolling-temperature appear in their most marked degree in the comparison of plates of different gauges. It is not customary to test the same heat in several sizes, but by long experience the manufacturer is able to judge the relative properties of each thickness. The heads of two widely-known plate-mills have given me as their estimate that, taking one-half inch as a basis, there will be the following changes in the physical properties for every increase of one-quarter inch in thickness:

"1. A decrease in ultimate strength of 1000 pounds per square inch.

"2. A decrease in elongation of 1 per cent. when measured in an 8-inch parallel section.

"3. A decrease in reduction of area of 2 per cent.

"W. R. Webster gives the same data on ultimate strength, but does not mention the relation of section to elongation.

"It is, therefore, plain that in the writing of specifications some allowance must be made for these conditions, since a requirement which is perfectly proper for a $\frac{3}{8}$-inch plate will be unreasonable for a $1\frac{1}{2}$-inch. Moreover, the effect is cumulative, since a harder steel must be used in making the thick plate, and this will tend to lessen the ductility rather than make up for the reduction caused by the larger section. In plates below $\frac{3}{8}$-inch in thickness it is also necessary to make allowances, since it is almost impossible to finish them at a high temperature, and the test will give a high ultimate strength and a low ductility.

"These conditions have now been officially recognized by the United States Government, for the rules of the Board of Supervising Inspectors, issued February 19, 1896, contain the following clause:

"'The sample must show, when tested, an elongation of at least 25 per cent. in a length of 2 inches for thickness up to $\frac{1}{4}$ inch, inclusive; and in a length of 4 inches for over $\frac{1}{4}$ to $\frac{7}{16}$, inclusive; and in a length of 8 inches for over $\frac{7}{16}$ to 1 inch, inclusive; and in a length of 6 inches, for all thicknesses over 1 inch.

"'It is to be hoped that constructive engineers will follow this example in recognizing the influence of causes over which the manufacturer has no control.' . . .

"The effect of sulphur upon the cold properties of steel has not been accurately determined, but it is quite certain that it is unimportant. In common practice the content varies from .02 to .10 per cent., and within these limits it seems to have no appreciable influence upon the elastic ratio, the elongation, or the reduction of area. It is more difficult to say that it does not alter the tensile strength, for a change of 1000 pounds per square inch can be caused by so many things that it is a bold venture to ascribe it to one variable. Webster has stated that sulphur probably increases the ultimate strength at the rate of 500 pounds per square inch for every .01 per cent. I am inclined to think his conclusion is not founded on sufficient premises, and shall try to prove this in Sections 161 and 163.

* *The Manufacture and Properties of Structural Steel*, New York, 1896, pp. 207, 208, 270, 284 to 287.

"In rivets, eyebars, and fire-box steel, the presence of sulphur is objectionable, for it will tend to create a coarse crystallization when the metal is heated to a high temperature, and reduce the strength and toughness of the steel. In other forms of structural material the effect of this element is probably of little importance. . . .

"The most comprehensive and systematic study of the physical formula of steel has been carried out by W. R Webster. He has used the long and laborious method of successive approximations, and by "cutting and trying" has found the effect of each element upon the ultimate strength, as well as the effect of the thickness and finishing temperature. The results are given by him as follows:

".01 per cent. of sulphur increases the tensile strength 500 pounds per square inch.

".01 per cent. of manganese has an effect which varies with each increment as follows, the values being expressed in pounds per square inch:

An Increase in Percentage	Gives an Increment of	Making a Total Increase in Strength Over Metal with no Manganese of
From .00 to .15	3,600	3,600
" .15 to .20	1,200	4,800
" .20 to .25	1,100	5,900
" .25 to .30	1,000	6,900
" .30 to .35	900	7,800
" .35 to .40	800	8,600
" .40 to .45	700	9,300
" .45 to .50	600	9,900
" .50 to .55	500	10,400
" .55 to .66	500	10,900
" .60 to .65	500	11,400

".01 per cent. of phosphorus has an effect which varies according to the amount of carbon present:

With .08 per cent. of	carbon it is		800	pounds per	square inch.
" .09	"	"	900	"	"
" .10	"	"	1000	"	"
" .11	"	"	1100	"	"
" .12	"	"	1200	"	"
" .13	"	"	1300	"	"
" .14	"	"	1400	"	"
" .15	"	"	1500	"	"
" .16	"	"	1500	"	"
" .17	"	"	1500	"	"

"Carbon is credited with a constant effect of 800 pounds for each .01 per cent.

"Mr. Webster has constructed, from these values, a table showing the strength of metal containing different proportions of carbon and phosphorus, from which, as a basis, the strength of a given steel may be found by allowing for the content of manganese and sulphur. This table presents a curious anomaly, as will be shown by the following excerpt:

Estimated Ultimate Strengths; Pounds per Square Inch, per Webster.

Carbon. Per Cent.	.07	.08	.09	.10	.11	.12	.13	.14	.15
Per Cent.									
P=.00........	40,350	41,150	41,950	42,750	43,550	44,350	45,150	45,950	46,750
P=.03........	42,750	43,550	44,650	45,750	46,850	47,950	49,050	50,150	51,250
P=.06........	45,150	45 950	47,350	48,750	50,150	51,550	52,950	54,350	55,750
P=.10........	48,350	49,150	50,950	52,750	54,550	56,350	58,150	59,950	61,750

"An examination of these figures reveals two absolutely irreconcilable conditions, for Mr. Webster takes as his starting point the dictum that carbon is a constant, and proceeds to construct a table in which it is not a constant at all, and in which it is not even constantly irregular. By his own calculation a steel of .06 per cent. phosphorus and .10 per cent. carbon is strengthened 1400 pounds by the addition of .01 per cent. of carbon, while with .10 per cent. phosphorus it is strengthened 1800 pounds by the same addition. Assuredly, this is not a constant effect. Moreover, carbon does not even have a constant effect with the same content of other metalloids, for, with .10 per cent. of phosphorus, an increase in carbon from .07 to .08 per cent. raises the strength 800 pounds, while an increase from .08 to .09 per cent. strengthens it 1800 pounds.

"It would be just as correct to conclude from these results that phosphorus is a constant and carbon a variable as to say that carbon is a constant and phosphorus a variable. The changing values which it would be necessary to assign to carbon to fulfill the first assumption would be no more arbitrary and hypothetical than the changing values assigned to phosphorus by Mr. Webster, or the changing values which he has assigned to manganese. Thus the table which has been given is entirely indecisive, since it can be translated into two diametrically opposite readings, and it must be acknowledged that one empirical formula is as good as another, provided the same answers are obtained from both.

"This curious contradiction of the premises by the conclusion can only arise from some erroneous hypothesis in the values assigned to the different elements, for in the construction of such equations it is plain that an error in one factor must be atoned for by an opposite and equal error in another factor. If this reasoning be true, then very little faith can be attached to the formula as an expression of fundamental laws, however accurately the mathematical results may coincide with observations.

"It is to be regretted that the earnest endeavor of Mr. Webster to write the physical formula should have been hampered by the necessity of working on sheared plates, which are finished under greater variations of temperature than angles or bars, and furthermore that these plates were of basic Bessemer steel, a material which would not be chosen for its regularity. By correcting for thickness and finishing temperature, Mr. Webster has shown that about 90 per cent. of the heats investigated came within 5000 pounds per square inch of what his equation calls for.

"This is a very satisfactory result, and it is not in a spirit of hypercriticism (for my own results, to be given later, display examples of the same character), but from a strictly scientific point of view, that attention is called to the very unpleasant corollary that one charge out of every ten does not give results within

5000 pounds. Some of these undoubtedly are vitiated by wrong chemical determinations, for the carbon was determined by color, and this gives only approximate results ; on others there might well be an error in estimating the finishing temperature ; on others there would be mistakes in measuring and testing ; while some pieces, perhaps, did actually show those peculiarities which we call abnormal, which are ascribed sometimes to oxide of iron, sometimes to nitrogen, and not infrequently to the devil, but which grow less numerous as we learn more of our art.

"I cannot believe that the complicated formula of Mr. Webster represents actual conditions, and the remainder of this chapter will attempt to show that a reasonably accurate empirical equation of steel may be written without the introduction of such manifold variations, and by the use of constant values for each element within the limits usually obtaining in structural metal. It will also be shown that the first increments of manganese do not add greatly to the strength of steel, since low-manganese metal is stronger than would be indicated by a formula that applies to steels containing higher percentages of this element."

I think Mr. Campbell's criticism is based upon an insufficient study of my table. He, moreover, ignores entirely the fact that a great deal of my work was on universal mill-plates.

As Mr. Campbell does not believe that my tables represent actual conditions, I would ask him if any one of the elements had a different effect per unit, due to a larger or smaller amount of the other elements present, what value would be arrived at by the method of least squares? Would it not be the average value of this element? If this is the case, why is he so positive that my values are not right, and that the effect of phosphorus is not greater as the carbon increases? Also, that the effects of the other elements are always the same per unit, no matter in what amount they may be present in the steels under consideration.

In other portions of his paper Mr. Campbell is not so positive in his statements on this point and others referred to above, as will be seen by the following:*

"It would seem, therefore, that the regularly increasing banefulness of phosphorus as the carbon is raised does not portray any change in nature, but that although the effect of the metalloid in lower steels is obscured, its character is the same. No line can be drawn that can be called the limit of safety, since no practical test has ever been devised which completely represents the effect of incessant tremor. For common structural material the critical content has been placed at .10 per cent. by general consent, but this is altogether too high for railroad-bridge work. All that can be said is that safety increases as phosphorus decreases, and the engineer may calculate just how much he is willing to pay for greater protection from accident. . . .

* *The Manufacture and Properties of Structural Steel,* New York, 1896, pp. 273, 291, 292.

"It is certain that carbon increases the strength of steel when present in small proportions, but that after a certain content is reached (say about 1 per cent.) there is no increase in cohesive power from a further addition. It will also be granted that this point is not a sudden break in the line, but that the effect of each unit of carbon decreases as it is approached. If this relation holds good throughout the whole series of alloys, then each successive increment of carbon will have a less effect from the starting-point of pure iron.

"It is also possible, for the same reasons, that every other metalloid will follow the same rule, so that the influence of each separate alloyed element will be represented by a curve. This may be an arc of a circle, or a parabola, or a cycloid, or a broken line; it may be different in degree or different in nature in the case of each element, and it may vary in degree or even in nature with changes in the proportions of the associated elements. But it will be assumed in this investigation that within the narrow limits of the divisions of the table the effect of a regular increase in the percentage of each metalloid would be represented by a straight line. In other words, that an increase of carbon from .20 to .21 per cent. gives the same increment in strength as an increase from .10 to .11 per cent."

It is hardly necessary to work out the cases referred to by Mr. Campbell, after the full explanations already given as to how my tables were constructed; but I have worked them out, and they are as follows:

Pure Iron.		Additions for Carbon.		Additions for Phosphorus			Difference.
34,750	+	800 × .10 C.	+	1,000 × .06 P.	=	48,750	
34,750	+	800 × .11 C.	+	1,100 × .06 P.	=	50,150	1,400
34,750	+	800 × .10 C.	+	1,000 × .10 P.	=	52,750	
34,750	+	800 × .11 C.	+	1,100 × .10 P.	=	54,550	1,800
34,750	+	800 × .07 C.	+	800 × .10 P.	=	48,350	
34,750	+	800 × .08 C.	+	800 × .10 P.	=	49,150	800
34,750	+	800 × .08 C.	+	800 × .10 P.	=	49,150	
34,750	+	800 × .09 C.	+	900 × .10 P.	=	50,950	1,800

In 1897 Mr. Cunningham* worked out, from Mr. Campbell's group of tests, values of 1000 pounds for each .01 per cent. of carbon and phosphorus, and 40,000 pounds for base. I quote as follows from my discussion† of his paper:

"The values he uses of 1000 pounds' increase for each .01 per cent. of carbon and phosphorus are the averages of Mr. Campbell's values for the same elements in acid and basic open-hearth steels, as is here shown:

* *Proc. Am. Soc. Civil Engineers*, xxiii., 231.

† *Trans. Am. Soc. Civil Engineers*, 1898, xl., 449 *et seq.*

Mr. Campbell's Values.

	Carbon.	Phosphorus.	Average of C and P.
Acid steel, ...	1,250	890	1,070
Basic steel, ...	950	1,050	1,000
Average for acid and basic, .	1,100	970	1,035

Using, for round numbers, 1000 pounds.

"In like manner, his base of 40,000 pounds is an average of the bases adopted by Mr. Campbell for acid and basic steels, with an addition for the average of .40 manganese, assumed by Mr. Cunningham to be in this steel. The following will show how this works out:

Mr. Campbell's Values.

	Base.	Addition for Mn.	Base Manganese.
Acid steel, ...	38,600	0	38,600
Basic steel, ...	37,430	3,400	40,830
Average for acid and basic, .	38,015	1,700	39,715

Using, for round numbers, 40,000 pounds."

"Mr. Cunningham says: 'The later investigations of Mr. Campbell are the most complete and scientific of any that have yet been undertaken in this line. With 3163 tests made upon 2 by ⅜-inch test-bars of a known and uniform condition, arranged in 272 groups of similar conditions as to strength and composition, Mr. Campbell has, by the method of least squares, arrived at the strengthening effect of the various components of steel,' notwithstanding the fact that Mr. Campbell, in his paper, admits that in applying the method of least squares, and using all of the elements, his results were not intelligible, and that first one element and then another had to be eliminated from his equations until he could obtain results that were applicable to the problem at hand. I do not, of course, know how familiar Mr. Cunningham and Mr. Campbell are with this method of least squares, but believe that the question is still an open one, as to whether or not it is the best method to use.

"Messrs. Cunningham and Campbell's results are based on figures obtained by grouping the original tests in accordance with their chemical composition, and taking the average results. I think they will find, upon further investigation of the subject, that it is much better to take individual cases, and not an average chemical composition, as this tends to mask the influence of the elements when existing in different proportions. In the application of any of these values to individual cases, tests in which the estimated ultimate strengths do not agree at all with the tensile tests will be found from time to time, and from such apparently abnormal cases more will be learned of the influence of each element, and of proper treatment in heating and rolling, than from the tests which agree with the estimated ultimate strengths.

"In order to compare Mr. Cunningham's method with mine, in its application to individual tests, I have taken the 408 tests given in my previous papers and worked out the estimated ultimate strength in each case by Mr. Cunningham's figures, and tabulated the results in the same general form as in my previous work. I have deducted the estimated ultimate strength from the actual ultimate

strength, and in the following tables my figures are, of course, the same as given in former papers. In this table the differences between the estimated and the actual ultimate strengths are subdivided, as noted in first column, and are recorded in the proper division in columns marked 'Cunningham' and 'Webster.' This table shows the necessity of considering other elements than carbon and phosphorus, in estimating the strength of steel from its chemical composition. The tests recorded in this table in the division marked plus over 10,000 pounds are not considered in the summary, as they are included in the division of plus over 5000 pounds, and are only given to show how much out of the way are the ultimate strengths calculated by Mr. Cunningham's formula when applied to individual tests of steel under 75,000 pounds tensile strength.

Webster's 408 *Tests.*

	Webster.	Cunningham.
+ over 10,000 pounds,	0	28
+ over 5000 pounds,	18	175
+ 4 to 5000 pounds,	18	53
+ 3 to 4000 pounds,	26	44
+ 2 to 3000 pounds,	35	50
+ 1 to 2000 pounds,	53	34
Within ± 1000 pounds,	106	44
— 1 to 2000 pounds,	54	5
— 2 to 3000 pounds,	41	2
— 3 to 4000 pounds,	28	1
— 4 to 5000 pounds,	18	0
— over 5000 pounds,	11	0
— over 10,000 pounds,	0	0

Summary.

Total + more than 1000 pounds,	150	356
Total — more than 1000 pounds,	152	8
Difference,	— 2	+ 348
Per cent		
Within 1000 pounds,	26.0	10.8
Within 2000 pounds,	52.2	20.3
Within 3000 pounds,	70.8	33.0
Within 4000 pounds,	84.1	44.1
Within 5000 pounds,	92.9	57.1

I have used Mr. Cunningham's values in preparing Tables XXI., XXII. and XXIII. of the present paper, which are similar to the other tables I have prepared, and will require no explanation.

By arranging in tables, constructed on some general form, the values of carbon, phosphorus and the other elements as given by different investigators, it is made an easy matter to

use the value of one element adopted by any given investigator with those of one or more elements as used by other investigators. In this way a much wider range of values can be covered, and indications sooner reached of the modifications or corrections required in value to meet any given condition of rolling. From the results that we have up to the present time, all that can be said is that we are no doubt on the right track, although there is much work to be done before we arrive at the final solution of the problem. I think that we should feel very much encouraged with the progress already made, and should renew the attack with increased vigor. The actual use of chemical data by the steel works in their ordinary everyday practice of grading the steel, and applying it to their orders, is much greater than is ordinarily supposed. This will in itself, in a few years, solve many of the points that are now giving us trouble.

In 1897 the Committee on Science and the Arts of the Franklin Institute in a report on my work said:

"The exact influence of the component elements on the physical characteristics of steel is probably largely determined by the condition in which the element occurs in the steel, and also by the inter-action of the various elements on each other, making the subject very complex. The most experienced physicists appear to entertain this view at the present time. It may therefore be an incorrect deduction, to broadly accept the values given by Mr. Webster as the constant strengthening effects of the elements he names. Nevertheless it is quite probable that the general tendency lies in the direction he indicates.

"The subject is one on which there has been considerable diversity of opinion, some observers disputing absolutely the strengthening influence of certain elements which Mr. Webster includes in his observation. A strong argument in his favor lies in the fact that, since the publication of his papers we know of no adverse criticism which weakens his conclusions. The work of Mr. Webster takes suitable rank amongst investigations of its character in a continuous series and according to a logical system, and (irrespective of the accuracy of his deductions) they constitute a valuable addition to the common store of knowledge."

The objection has been raised that the chemist only gives the total carbon present in the steel, and not the condition in which it exists, and that we cannot expect to predict from this total carbon what its physical effect will be, as in one case we may have much more of the hardening carbon present than in another for the same total carbon reported. This objection is not as important as it seems; for the form of the carbon present depends largely on the heat-treatment, and that is again modi-

fied by work of rolling; therefore, if we take any given grade of steel, and by experiment determine the physical effect of different heat-treatments in connection with work, we have the direct answer instead of waiting for the proportion of hardening carbon present to be given by the chemist or the microscopist. We know to-day that as carbon increases, the differences due to heat-treatment or finishing-temperature in rolling are much greater than in the lower carbon steels. This is no doubt due to the greater change in the form of the carbon present. It calls for a little more leeway between the high and low limits of ultimate strength in specifications, and much closer vigilance as to heating- and finishing-temperatures in rolling the higher steel. Microscopic examinations will, of course, be of the greatest service in this connection, and will give us definite information on many points that are now in doubt. The microscopists have not as yet seriously taken up the solution of this problem, and to-day we do not know how the different fractures of steel, due to well-known changes from heat-treatment, would appear under the microscope, and we could not recognize them. Then, again, the grain referred to by Mr. Sauveur and others is not the same grain that we refer to and are familiar with in ordinary practice. I look forward to great results from the microscopical investigation co-operating with chemical analyses and the study of heat-treatment in connection with work.

In this practical age it will be asked, "What is the actual value of all this, and is it worth while to bother any more about it?" The answer is that already the steel-maker depends more and more every year on the very points that we have been discussing, and the engineer should know what he is doing when he uses both chemical and physical limits in his specifications. Is it to be wondered at that to-day we have specifications in which the chemical requirements do not at all agree with the ultimate strengths specified? This in the ordinary home-orders may make very little difference, as it is an easy matter to investigate and make proper modifications required, but in specifications from foreign railroads or engineers at a distance, for material on export orders, it is a very different matter. There is no time to refer questions of this kind, and the inspector is sometimes forced to condemn material that he knows

from experience is all right. It may give better physical tests than were called for but not conform to the chemical requirements in the particular specifications in question, though it would meet both the physical and chemical requirements of our leading engineers and railway companies in this country. Some of these foreign specifications are so worded, and the requirements are such, that it is not safe for an American manufacturer to bid on the orders. This may be due in part to their not understanding all of the conditions under which the tests are to be made. In other cases it seems as though modifications must have been made on previous orders, either in the chemical requirements or physical tests called for, as no steel was ever made that would meet all the requirements of chemical and physical tests in some such specifications.

Many export-orders for rails call for phosphorus not over .06 per cent., while the sulphur is allowed a much higher limit, or not specified at all. This low phosphorus cuts out several of our largest steel-works, whose material will meet all of the physical-tests called for, being low in sulphur and under .10 per cent. phosphorus, and giving entire satisfaction to our leading engineers and railroad-companies. The foreign manufacturers of basic Bessemer steel and low-phosphorus acid Bessemer steel are working the low-phosphorus clause for all it is worth, just as our manufacturers would do under similar circumstances; but we, as engineers, want to know if there is any good reason for this distinction, which is made on account of local conditions, and if the high sulphur is not as undesirable as the phosphorus would be up to the limit of .10 per cent. usually allowed in this country?

This is a matter that should be thoroughly discussed and on which opinions should be secured from our leading engineers; for it is of no use for the manufacturers to give their opinion unsupported by the experience of users of their rails. It is just here that the effect of the different elements plays an important part. Our manufacturers know from experience that as their phosphorus is a little higher than the limit of .06 per cent. referred to, they must keep the other elements inside of given limits to produce the best results in the finished rails. This is done successfully, and a rail is produced that will meet the requirements of the drop-test and give satisfaction in use. I

do not wish to be understood as advocating high-phosphorus, but my position is, that rails which will meet all of the varying conditions in this country will meet all that is required of them when put in use in other countries. If one examines the methods of manufacture of rails in this country and those in other countries, then visits the countries where our rails are being introduced, and gets accurate information as to the service they are giving, he will be convinced that there is a great future for America in the export-business in rails.

The conditions with regard to the material for bridges are about the same. Foreign specifications, in many cases, provide that only acid open-hearth steel can be used. This has already prevented our bridge-builders from bidding upon thousands of tons of bridge-work, although in some cases they have accepted the requirement of acid open-hearth steel, and have taken the orders. In other cases they have explained that our Government is using the basic open-hearth steel; that in the manufacture of it the works here start with an iron very much lower in phosphorus than is ordinarily used, and that the resulting steel is much more uniform. This has had a great deal of weight, and the use of basic steel has been allowed in some cases. In other cases new specifications have been adopted after investigation in accordance with the best American practice, which allows the use of either basic or acid open-hearth steel. I refer to this matter at length, as it is of the greatest importance to our engineers and manufacturers, as well as to all others who wish to see our export-trade developed.

The continued discussion on the physics of steel is broad enough to take up any specifications on home- or export-orders that may be brought forward; the uniform methods of testing and methods of manufacture and rolling that will improve the finished products. It is of the greatest importance to establish the quality of American material abroad, and to have on record all facts regarding the same, both from the engineers' side and the manufacturers', and I know of no better channel for doing this than the American Institute of Mining Engineers.

TABLE I.—*Webster's Values for Carbon, Carbon + Base of 38,000 Lbs., and Carbon + Base of 34,750 Lbs.*

Carbon.	Carbon Only.	Carbon + 38,000 Lbs.	Carbon + 34,750 Lbs.
Per Cent.	Lbs.	Lbs.	Lbs.
.05	4,000	42,000	38,750
.06	4,800	42,800	39,550
.07	5,600	43,600	40,350
.08	6,400	44,400	41,150
.09	7,200	45,200	41,950
.10	8,000	46,000	42,750
.11	8,800	46,800	43,550
.12	9,600	47,600	44,350
.13	10,400	48,400	45,150
.14	11,200	49,200	45,950
.15	12,000	50,000	46,750
.16	12,800	50,800	47,550
.17	13,600	51,600	48,350
.18	14,400	52,400	49,150
.19	15,200	53,200	49,950
.20	16,000	54,000	50,750
.21	16,800	54,800	51,550
.22	17,600	55,600	52,350
.23	18,400	56,400	53,150
.24	19,200	57,200	53,950
.25	20,000	58,000	54,750
.26	20,800	58,800	55,550
.27	21,600	59,600	56,350
.28	22,400	60,400	57,150
.29	23,200	61,200	57,950
.30	24,000	62,000	58,750
.31	24,800	62,800	59,550
.32	25,600	63,600	60,350
.33	26,400	64,400	61,150
.34	27,200	65,200	61,950
.35	28,000	66,000	62,750
.36	28,800	66,800	63,550
.37	29,600	67,600	64,350
.38	30,400	68,400	65,150
.39	31,200	69,200	65,950
.40	32,000	70,000	66,750
.41	32,800	70,800	67,550
.42	33,600	71,600	68,350
.43	34,400	72,400	69,150
.44	35,200	73,200	69,950
.45	36,000	74,000	70,750
.46	36,800	74,800	71,550
.47	37,600	75,600	72,350
.48	38,400	76,400	73,150
.49	39,200	77,200	73,950
.50	40,000	78,000	74,750
.51	40,800	78,800	75,550
.52	41,600	79,600	76,350
.53	42,400	80,400	77,150
.54	43,200	81,200	77,950
.55	44,000	82,000	78,750
.56	44,800	82,800	79,550
.57	45,600	83,600	80,350
.58	46,400	84,400	81,150
.59	47,200	85,200	81,950
.60	48,000	86,000	82,750

TABLE II.—*Webster's Values for Phosphorus in the Presence of .05, .06, .07 and .08 Per Cent. Carbon.*

Per Cent.	.000	.001	.002	.003	.004	.005	.006	.007	.008	.009
	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.
.00	000	80	160	240	320	400	480	560	640	720
.01	800	880	960	1,040	1,120	1,200	1,280	1,360	1,440	1,520
.02	1,600	1,680	1,760	1,840	1,920	2,000	2,080	2,160	2,240	2,320
.03	2,400	2,480	2,560	2,640	2,720	2,800	2,880	2,960	3,040	3,120
.04	3,200	3,280	3,360	3,440	3,520	3,600	3,680	3,760	3,840	3,920
.05	4,000	4,080	4,160	4,240	4,320	4,400	4,480	4,560	4,640	4,720
.06	4,800	4,880	4,960	5,040	5,120	5,200	5,280	5,360	5,440	5,520
.07	5,600	5,680	5,760	5,840	5,920	6,000	6,080	6,160	6,240	6,320
.08	6,400	6,480	6,560	6,640	6,720	6,800	6,880	6,960	7,040	7,120
.09	7,200	7,280	7,360	7,440	7,520	7,600	7,680	7,760	7,840	7,920
.10	8,000	8,080	8,160	8,240	8,320	8,400	8,480	8,560	8,640	8,720

TABLE III.—*Webster's Values for Phosphorus in the Presence of .09 Per Cent. Carbon.*

Per Cent.	.000	.001	.002	.003	.004	.005	.006	.007	.008	.009
	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.
.00	000	90	180	270	360	450	540	630	720	810
.01	900	990	1,080	1,170	1,260	1,350	1,440	1,530	1,620	1,710
.02	1,800	1,890	1,980	2,070	2,160	2,250	2,340	2,430	2,520	2,610
.03	2,700	2,790	2,880	2,970	3,060	3,150	3,240	3,330	3,420	3,510
.04	3,600	3,690	3,780	3,870	3,960	4,050	4,140	4,230	4,320	4,410
.05	4,500	4,590	4,680	4,770	4,860	4,950	5,040	5,130	5,220	5,310
.06	5,400	5,490	5,580	5,670	5,760	5,850	5,940	6,030	6,120	6,210
.07	6,300	6,390	6,480	6,570	6,660	6,750	6,840	6,930	7,020	7,110
.08	7,200	7,290	7,380	7,470	7,560	7,650	7,740	7,830	7,920	8,010
.09	8,100	8,190	8,280	8,370	8,460	8,550	8,640	8,730	8,820	8,910
.10	9,000	9,090	9,180	9,270	9,360	9,450	9,540	9,630	9,720	9,810

TABLE IV.—*Webster's Values for Phosphorus in the Presence of .10 Per Cent. Carbon.*

Per Cent.	.000	.001	.002	.003	.004	.005	.006	.007	.008	.009
	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs	Lbs	Lbs.	Lbs	Lbs.
.00	000	100	200	300	400	500	600	700	800	900
.01	1,000	1,100	1,200	1,300	1,400	1,500	1,600	1,700	1,800	1,900
.02	2,000	2,100	2,200	2,300	2,400	2,500	2,600	2,700	2,800	2,900
.03	3,000	3,100	3,200	3,300	3,400	3,500	3,600	3,700	3,800	3,900
.04	4,000	4,100	4,200	4,300	4,400	4,500	4,600	4,700	4,800	4,900
.05	5,000	5,100	5,200	5,300	5,400	5,500	5,600	5,700	5,800	5,900
.06	6,000	6,100	6,200	6,300	6,400	6,500	6,600	6,700	6,800	6,900
.07	7,000	7,100	7,200	7,300	7,400	7,500	7,600	7,700	7,800	7,900
.08	8,000	8,100	8,200	8,300	8,400	8,500	8,600	8,700	8,800	8,900
.09	9,000	9,100	9,200	9,300	9,400	9,500	9,600	9,700	9,800	9,900
.10	10,000	10,100	10,200	10,300	10,400	10,500	10,600	10,700	10,800	10,900

TABLE V.—*Webster's Values for Phosphorus in the Presence of .11 Per Cent. Carbon.*

Per Cent.	.000	.001	.002	.003	.004	.005	.006	.007	.008	.009
	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.
.00	000	110	220	330	440	550	660	770	890	990
.01	1,100	1,210	1,320	1,430	1,540	1,650	1,760	1,870	1,980	2,090
.02	2,200	2,320	2,420	2,530	2,640	2,750	2,860	2,970	3,080	3,190
.03	3,300	3,400	3,520	3,630	3,740	3,850	3,960	4,070	4,180	4,290
.04	4,400	4,540	4,620	4,730	4,840	4,950	5,060	5,170	5,280	5,390
.05	5,500	5,610	5,720	5,830	5,940	6,050	6,160	6,270	6,380	6,490
.06	6,600	6,710	6,820	6,930	7,040	7,150	7,260	7,370	7,480	7,590
.07	7,700	7,810	7,920	8,030	8,140	8,250	8,360	8,470	8,580	8,690
.08	8,800	8,910	9,020	9,130	9,240	9,350	9,460	9,570	9,680	9,790
.09	9,900	10,010	10,120	10,230	10,340	10,450	10,560	10,670	10,780	10,890
.10	11,000	11,110	11,220	11,330	11,440	11,550	11,660	11,770	11,880	11,990

TABLE VI.—*Webster's Values for Phosphorus in the Presence of .12 Per Cent. Carbon.*

Per Cent.	.000	.001	.002	.003	.004	.005	.006	.007	.008	.009
	Lbs	Lbs	Lbs.	Lbs	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.
.00	000	120	240	360	480	600	720	840	960	1,080
.01	1,200	1,320	1,440	1,560	1,680	1,800	1,920	2,040	2,160	2,280
.02	2,400	2,520	2,640	2,760	2,880	3,000	3,120	3,240	3,360	3,480
.03	3,600	3,720	3,840	3,960	4,080	4,200	4,320	4,440	4,560	4,680
.04	4,800	4,920	5,040	5,160	5,280	5,400	5,520	5,640	5,760	5,880
.05	6,000	6,120	6,240	6,360	6,480	6,600	6,720	6,840	6,960	7,080
.06	7,200	7,320	7,440	7,560	7,680	7,800	7,920	8,040	8,160	8,280
.07	8,400	8,520	8,640	8,760	8,880	9,000	9,120	9,240	9,360	9,480
.08	9,600	9,720	9,840	9,960	10,080	10,200	10,320	10,440	10,560	10,680
.09	10,800	10,920	11,040	11,160	11,280	11,400	11,520	11,640	11,760	11,880
.10	12,000	12,120	12,240	12,360	12,480	12,600	12,720	12,840	12,960	13,080

TABLE VII.—*Webster's Values for Phosphorus in the Presence of .13 Per Cent. Carbon.*

Per Cent.	.000	.001	.002	.003	.004	.005	006	.007	.008	.009
	Lbs.	Lbs.	Lbs	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.
.00	000	130	260	390	520	650	780	910	1,040	1,170
.01	1,300	1,430	1,560	1,690	1,820	1,950	2,080	2,210	2,340	2,470
.02	2,600	2,730	2,860	2,990	3,120	3,250	3,380	3,510	3,640	3,770
.03	3,900	4,030	4,160	4,290	4,420	4,550	4,680	4,810	4,940	5,070
.04	5,200	5,330	5,460	5,590	5,720	5,850	5,980	6,110	6,240	6,370
.05	6,500	6,630	6,760	6,890	7,020	7,150	7,280	7,410	7,540	7,670
.06	7,800	7,930	8,060	8,190	8,320	8,450	8,580	8,710	8,840	8,970
.07	9,100	9,230	9,360	9,490	9,620	9,750	9,880	10,010	10,140	10,270
.08	10,400	10,530	10,660	10,790	10,920	11,050	11,180	11,310	11,440	11,570
.09	11,700	11,830	11,960	12,090	12,220	12,350	12,480	12,610	12,740	12,870
.10	13,000	13,130	13,260	13,390	13,520	13,650	13,780	13,910	14,040	14,170

TABLE VIII.—*Webster's Values for Phosphorus in the Presence of .14 Per Cent. Carbon.*

Per Cent.	.000	.001	.002	.003	.004	.005	.006	.007	.008	.009
	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.
.00	000	140	280	420	560	700	840	980	1,020	1,260
.01	1,400	1,540	1,680	1,820	1,960	2,100	2,240	2,380	2,520	2,660
.02	2,800	2,940	3,080	3,220	3,360	3,500	3,640	3,780	3,920	4,060
.03	4,200	4,340	4,480	4,620	4,760	4,900	5,040	5,180	5,320	5,460
.04	5,600	5,740	5,880	6,020	6,160	6,300	6,440	6,580	6,720	6,860
.05	7,000	7,140	7,280	7,420	7,560	7,700	7,840	7,980	8,120	8,260
.06	8,400	8,540	8,680	8,820	8,960	9,100	9,240	9,380	9,520	9,660
.07	9,800	9,940	10,080	10,220	10,360	10,500	10,640	10,780	10,920	11,060
.08	11,200	11,340	11,480	11,620	11,760	11,900	12,040	12,180	12,320	12,460
.09	12,600	12,740	12,880	13,020	13,160	13,300	13,440	13,580	13,720	13,860
.10	14,000	14,140	14,280	14,420	14,560	14,700	14,840	14,980	15,120	15,260

TABLE IX.—*Webster's Values for Phosphorus in the Presence of .15 Per Cent. Carbon and Over.*

Per Cent.	.000	.001	.002	.003	.004	.005	.006	.007	.008	.009
	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.
.00	000	150	300	450	600	750	900	1,050	1,200	1,350
.01	1,500	1,650	1,800	1,950	2,100	2,250	2,400	2,550	2,700	2,850
.02	3,000	3,150	3,300	3,450	3,600	3,750	3,900	4,050	4,200	4,350
.03	4,500	4,650	4,800	4,950	5,100	5,250	5,400	5,550	5,700	5,850
.04	6,000	6,150	6,300	6,450	6,600	6,750	6,900	7,050	7,200	7,350
.05	7,500	7,650	7,800	7,950	8,100	8,250	8,400	8,550	8,700	8,850
.06	9,000	9,150	9,300	9,450	9,600	9,750	9,900	10,050	10,200	10,350
.07	10,500	10,650	10,800	10,950	11,100	11,250	11,400	11,550	11,700	11,850
.08	12,000	12,150	12,300	12,450	12,600	12,750	12,900	13,050	14,200	14,350
.09	13,500	13,650	13,800	13,950	14,100	14,250	14,400	14,550	14,700	14,850
.10	15,000	15,150	15,300	15,450	15,600	15,750	15,900	16,050	16,200	16,350

TABLE X.—*Webster's Values for Phosphorus in the Presence of .05 to .15 Per Cent. Carbon, Inclusive.*

Per Cent.	Carbon .05, .06, .07, .08.	Carbon .09	Carbon .10	Carbon .11	Carbon .12	Carbon .13	Carbon .14	Carbon .15 and Over.
P. .000	0	0	0	0	0	0	0	0
" .005 ..	400	450	500	550	600	650	700	750
" .01 ...	800	900	1,000	1,100	1,200	1,300	1,400	1,500
" .015 ..	1,200	1,350	1,500	1,650	1,800	1,950	2,100	2,250
" .02	1,600	1,800	2,000	2,200	2,400	2,600	2,800	3,000
" .025.....	2,000	2,250	2,500	2,750	3,000	3,250	3,500	3,750
" .03	2,400	2,700	3,000	3,300	3,600	3,900	4,200	4,500
" .035. ...	2,800	3,150	3,500	3,850	4,200	4,550	4,900	5,250
" .04 . .	3,200	3,600	4,000	4,400	4,800	5,200	5,600	6,000
" .045.....	3,600	4,050	4,500	4,950	5,400	5,850	6,300	6,750
" .05	4,000	4,500	5,000	5,500	6,000	6,500	7,000	7,500
" 055 .	4,400	4,950	5,500	6,050	6,600	7,150	7,700	8,250
" .06	4,800	5,400	6,000	6,600	7,200	7,800	8,400	9,000
" .065 .	5,200	5,850	6,500	7,150	7,800	8,450	9,100	9,750
" .07	5,600	6,300	7,000	7,700	8,400	9,100	9,800	10,500
" .075 . .	6,000	6,750	7,500	8,250	9,000	9,750	10,500	11,250
" .08 .	6,400	7,200	8,000	8,800	9,600	10,400	11,200	12,000
" .085......	6,800	7,650	8,500	9,350	10,200	11,050	11,900	12,750
" .09	7,200	8,100	9,000	9,900	10,800	11,700	12 600	13,500
" .095	7,600	8,550	9,500	10,450	11,400	12,350	13,300	14,250
" .10 . . .	8,000	9,000	10,000	11,000	12,000	13,000	14,000	15,000
.001 P =...	80 lbs.	90 lbs.	100 lbs.	110 lbs.	120 lbs.	130 lbs.	140 lbs.	150 lbs.

TABLE XI.—*Estimated Ultimate Strengths by Webster's Values.*

Base = 38,000 lbs. per square inch for iron.
Increase of 800 lbs. for each .01 carbon.
" 800 lbs. to 1,500 lbs. for each .01 phosphorus, as explained.

Carbon.	.06	.07	.08	.09	.10	.11	.12	.13	.14	.15	.16	.17	.18	.19	.20	.21	.22	.23	.24	.25
Phos. .000.....	42,800	43,600	44,400	45,200	46,000	46,800	47,600	48,400	49,200	50,000	50,800	51,600	52,400	53,200	54,000	54,800	55,600	56,400	57,200	58,000
" .005.....	43,200	44,000	44,800	45,650	46,500	47,350	48,200	49,050	49,900	50,750	51,550	52,350	53,150	53,950	54,750	55,550	56,350	57,150	57,950	58,750
" .01	43,600	44,400	45,200	46,100	47,000	47,900	48,800	49,700	50,600	51,500	52,300	53,100	53,900	54,700	55,500	56,300	57,100	57,900	58,700	59,500
" .015.. ..	44,000	44,800	45,600	46,550	47,500	48,450	49,400	50,350	51,300	52,250	53,050	53,850	54,650	55,450	56,250	57,050	57,850	58,650	59,450	60,250
" .02	44,400	45,200	46,000	47,000	48,000	49,000	50 000	51,000	52,000	53,000	53,800	54,600	55,400	56,200	57,000	57,800	58,600	59,400	60,200	61,000
" .025.....	44,800	45,600	46,400	47,450	48,500	49,550	50,600	51,650	52,700	53,750	54,550	55,350	56,150	56,950	57,750	58,550	59.350	60,150	60,950	61,750
" .03	45,200	46,000	46,800	47,900	49,000	50,100	51,200	52,300	53,400	54,500	55,300	56,100	56,900	57,700	58,500	59,300	60,100	60,900	61,700	62,500
" .035.. ..	45,600	46,400	47,200	48,350	49,500	50,650	51,800	52,950	54,100	55,250	56,050	56,850	57,650	58,450	59,250	60,050	60,850	61,650	62,450	63,250
" .04	46,000	46,800	47,600	48,800	50,000	51,200	52,400	53,600	54,800	56,000	56,800	57,600	58,400	59,200	60,000	60,800	61,600	62,400	63,200	64,000
" .045.....	46,400	47,200	48,000	49,250	50,500	51,750	53,000	54,250	55,500	56,750	57,550	58,350	59,150	59,950	60,750	61,550	62,350	63,150	63,950	64,750
" .05	46,800	47,600	48,400	49,700	51,000	52,300	53,600	54,900	56,200	57,500	58,300	59,100	59,900	60,700	61,500	62,300	63,100	63,900	64,700	65,500
" .055 ...	47,200	48,000	48,800	50,150	51,500	52,850	54,200	55,550	56,900	58,250	59,050	59,850	60,650	61,450	62,250	63,050	63,850	64,650	65,450	66,250
" .06	47,600	48,400	49,200	50,600	52,000	53,400	54,800	56,200	57,600	59,000	59,800	60,600	61,400	62,200	63,000	63,800	64,600	65,400	66,200	67,000
" .065... .	48,000	48,800	49,600	51,050	52,500	53,950	55,400	56,850	58,300	59,750	60,550	61,350	62,150	62,950	63,750	64,550	65,350	66,150	66,950	67,750
" .07	48,400	49,200	50,000	51,500	53,000	54,500	56,000	57,500	59,000	60,500	61,300	62,100	62,900	63,700	64,500	65,300	63,100	66,900	67,700	68,500
" .075.....	48,800	49,600	50,400	51,950	53,500	55,050	56,600	58,150	59,700	61,250	62,050	62,850	63,650	64,450	65,250	66,050	66,850	67,650	68,450	69,250
" .08	49,200	50,000	50,800	52,400	54,000	55,600	57,200	58,800	60,400	62,000	62,800	63,600	64,400	65,200	66,000	66,800	67,600	68,400	69,200	70,000
" .085.....	49,600	50,400	51,200	52,850	54,500	56,150	57,800	59,450	61,100	62,750	63,550	64,350	65,150	65,950	66,750	67,550	68,350	69,150	69,950	70,750
" .09	50,000	50,800	51,600	53,300	55,000	56,700	58,400	60,100	61,800	63,500	64 300	65,100	65,900	66,700	67,500	68,300	69,100	69,900	70,700	71,500
" .095.....	50,400	51,200	52,000	53,750	55,500	57,250	59 000	60,750	62,500	64,250	65,050	65,850	66,650	67,450	68,250	69,050	69,850	70,650	71,450	72,250
" .10	50,800	51,600	52,400	54,200	56,000	57,800	59,600	61,400	63,200	65,000	65,800	66,600	67,400	68,200	69,000	69,800	70 600	71,400	72,200	73,000
.001 Phos.=..	80 lbs.	80 lbs.	80 lbs.	90 lbs.	100 lbs.	110 lbs.	120 lbs	130 lbs.	140 lbs	150 lbs.	150 lbs.	150 lbs.	150 lbs.	150 lbs.	150 lbs.	150 lbs.	150 lbs.	150 lbs.	150 lbs.	150 lbs.

TABLE XII.—*Estimated Ultimate Strengths by Webster's Values.*

Base = 34,750 lbs. per square inch for iron.
Increase of 800 lbs. for each .01 per cent. of carbon.
" 800 to 1,500 lbs. for each .01 per cent. of phosphorus, as explained.

Carbon =	.06	.07	.08	.09	.10	.11	.12	13	.14	.15	.16	.17	.18	.19	.20	.21	.22	.23	.24	.25
Phos. .000	39,550	40,350	41,150	41,950	42,750	43,550	44,350	45,150	45,950	46,750	47,550	48,350	49,150	49,950	50,750	51,550	52,350	53,150	53,950	54.750
" .005	39,950	40,750	41,550	42,400	43,250	44,100	44 950	45,800	46,650	47,500	48,300	49,100	49,900	50,700	51,500	52,300	53,100	53,900	54,700	55,500
" .01	40,350	41,150	41,950	42,850	43,750	44,650	45 550	46,450	47,350	48,250	49,050	49,850	50,650	51,450	52,250	53,050	53,850	54,650	55,450	56,250
" .015	40,750	41,550	42,350	43,300	44,250	45,200	46,150	47,100	48,050	49,000	49,800	50,600	51,400	52,200	53,000	53,800	54,600	55,400	56,200	57,000
" .02	41,150	41,950	42,750	43,750	44,750	45,750	46,750	47,750	48,750	49,750	50,550	51,350	52,150	52,950	53,750	54,550	55,350	56,150	56,950	57,750
" .025	41,550	42,350	43,150	44,200	45,250	46,300	47,350	48,400	49,450	50,500	51,300	52,100	52,900	53,700	54,500	55,300	56,100	56,900	57,700	58,500
" .03	41,950	42,750	43,550	44,650	45,750	46,850	47 950	49,050	50,150	51,250	52,050	52,850	53,650	54,450	55,250	56,050	56,850	57,650	58,450	59,250
" .035	42,350	43,150	43,950	45,100	46,250	47,400	48,550	49,700	50,850	52,000	52,800	53,600	54,400	55,200	56,000	56,800	57,600	58,400	59,200	60,000
" .04	42,750	43,550	44,350	45,550	46,750	47,950	49,150	50,350	51,550	52,750	53,550	54,350	55,150	55,950	56,750	57,550	58,350	59,150	59 950	60,750
" .045	43,150	43,950	44,750	46,000	47,250	48,500	49,750	51,000	52,250	53,500	54,300	55,100	55,900	56,700	57,500	58,300	59,100	59,900	60,700	61,500
" .05	43,550	44,350	55,150	46,450	47,750	49,050	50,350	51,650	52,950	54,250	55,050	55,850	56,650	57,450	58,250	59,050	59,850	60.650	61,450	62,250
" .055 ...	43,950	44,750	45,550	46,900	48,250	49,600	50,950	52,300	53,650	55,000	55,800	56,600	57,400	58,200	59,000	59,800	60,600	61,400	62,200	63,000
" .06 ..	44,350	45,150	45,950	47,350	48,750	50,150	51,550	52,950	54,350	55,750	56,550	57,350	58,150	58,950	59,750	60,550	61,350	62,150	62,950	63,750
" .065	41,750	45,550	46,350	47,800	49,250	50,700	52,150	53,600	55,050	56,500	57,300	58,100	58,900	59,700	60,500	61,300	62,100	62,900	63,700	64,500
" .07	45,150	45,950	46,750	48,250	49,750	51,250	52,750	54,250	55,750	57,250	58,050	58,850	59,650	60,450	61,250	62,050	62,850	63,650	64,450	65,250
" .075 ...	45,550	46,350	47,150	48,700	50,250	51,800	53,350	54,900	56,450	58,000	58,800	59,600	60,400	61,200	62,000	62,800	63,600	64,400	65,200	66,000
" .08	45,950	46,750	47,550	49,150	50,750	52,350	53,950	55,550	57,150	58,750	59,550	60,350	61,150	61,950	62,750	63,550	64,350	65,150	65,950	66,750
" .085	46,350	47,150	47,950	49,600	51,250	52,900	54,550	56,200	57,850	59,500	60,300	61,100	61,900	62,700	63,500	64.300	65,100	65,900	66,700	67,500
" .09	46,750	47,550	38,350	50,050	51,750	53,450	55,150	56 850	58,550	60,250	61,050	61,850	62,650	63,450	64,250	65,050	65,850	66,650	67,450	68.250
" .095 ..	47,150	47,950	48,750	50,500	52,250	54,000	55,750	57,500	59,250	61,000	61,800	62.600	63,400	64,200	65,000	65,800	66,600	67.400	68,200	69,000
" .10	47,550	48,350	49,150	50,950	52,750	54,550	56,350	58,150	59,950	61,750	62,550	63,350	64,150	64,950	65,750	66,550	67,330	68,150	68,950	69,750
.001 Phos. =..	80 lbs.	80 lbs.	80 lbs.	90 lbs.	100 lbs.	110 lbs.	120 lbs	130 lbs.	140 lbs.	150 lbs	150 lbs.	150 lbs.	150 lbs.	150 lbs.	150 lbs.	150 lbs.	150 lbs.	150 lbs.	150 lbs.	150 lbs.

TABLE XIII.—*Webster's Additions for Manganese.*

Mn. Per Cent.	Lbs.	Mn. Per Cent.	Lbs.	Mn. Per Cent.	Lbs.	Mn. Per Cent.	Lbs.
		.26	6,100	.51	10,000	.76	12,500
		.265	6,200	.515	10,050	.765	12,550
		.27	6,300	.52	10,100	.77	12,600
		.275	6,400	.525	10,150	.775	12,650
		.28	6,500	.53	10,200	.78	12,700
		.285	6,600	.535	10,250	.785	12,750
		.29	6,700	.54	10,300	.79	12,800
		.295	6,800	.545	10,350	.795	12,850
		.30	6,900	.55	10,400	.80	12,900
		.305	6,990	.555	10,450	.805	12,950
		.31	7,080	.56	10,500	.81	13,000
		.315	7,170	.565	10,550	.815	13,050
		.32	7,260	.57	10,600	.82	13,100
		.325	7,350	.575	10,650	.825	13,150
		.33	7,440	.58	10,700	.83	13,200
		.335	7,530	.585	10,750	.835	13,250
		.34	7,620	.59	10,800	.84	13,300
		.345	7,710	.595	10,850	.845	13,350
		.35	7,800	.60	10,900	.85	13,400
		.355	7,880	.605	10,950	.855	13,450
		.36	7,960	.61	11,000	.86	13,500
		.365	8,040	.615	11,050	.865	13,550
		.37	8,120	.62	11,100	.87	13,600
		.375	8,200	.625	11,150	.875	13,650
		.38	8,280	.63	11,200	.88	13,700
		.385	8,360	.635	11,250	.885	13,750
		.39	8,440	.64	11,300	.89	13,800
		.395	8,520	.645	11,350	.895	13,850
.15	3,600	.40	8,600	.65	11,400	.90	13,900
.155	3,720	.405	8,670	.655	11,450	.905	13,950
.16	3,840	.41	8,740	.66	11,500	.91	14,000
.165	3,960	.415	8,810	.665	11,550	.915	14,050
.17	4,080	.42	8,880	.67	11,600	.92	14,100
.175	4,200	.425	8,950	.675	11,650	.925	14,150
.18	4,320	.43	9,020	.68	11,700	.93	14,200
.185	4,440	.435	9,090	.685	11,750	.935	14,250
.19	4,560	.44	9,160	.69	11,800	.94	14,300
.195	4,680	.445	9,230	.695	11,850	.945	14,350
.20	4,800	.45	9,300	.70	11,900	.95	14,400
.205	4,910	.455	9,360	.705	11,950	.955	14,450
.21	5,020	.46	9,420	.71	12,000	.96	14,500
.215	5,130	.465	9,480	.715	12,050	.965	14,550
.22	5,240	.47	9,540	.72	12,100	.97	14,600
.225	5,350	.475	9,600	.725	12,150	.975	14,650
.23	5,460	.48	9,660	.73	12,200	.98	14,700
.235	5,570	.485	9,720	.735	12,250	.985	14,750
.24	5,680	.49	9,780	.74	12,300	.99	14,800
.245	5,790	.495	9,840	.745	12,350	.995	14,850
.25	5,900	.50	9,900	.75	12,400	1.00	14,900
.255	6,000	.505	9,950	.755	12,450		

TABLE XIV.—*Webster's Additions for Sulphur.* (.01 *Per Cent. Sulphur* = 500 *Lbs.*)

Sulphur. Per Cent.	.0	.001	.002	.003	.004	.005	.006	.007	.008	.009
	Lbs	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.
.0	000	50	100	150	200	250	300	350	400	450
.01..........	500	550	600	650	700	750	800	850	900	950
.02..........	1,000	1,050	1,100	1,150	1,200	1,250	1,300	1,350	1,400	1,450
.03..........	1,500	1,550	1,600	1,650	1,700	1,750	1,800	1,850	1,900	1,950
.04..........	2,000	2,050	2,100	2,150	2,200	2,250	2,300	2,350	2,400	2,450
.05..........	2,500	2,550	2,600	2,650	2,700	2,750	2,800	2,850	2,900	2,950
.06..........	3,000	3,050	3,100	3,150	3,200	3,250	3,300	3,350	3,400	3,450
.07..........	3,500	3,550	3,600	3,650	3,700	3,750	3,800	3,850	3,900	3,950
.08..........	4,000	4,050	4,100	4,150	4,200	4,250	4,300	4,350	4,400	4,450
.09..........	4,500	4,550	4,600	4,650	4,700	4,750	4,800	4,850	4,900	4,950
.10..........	5,000	5,050	5,100	5,150	5,200	5,250	5,300	5,350	5,400	5,450

TABLE XV.—*Campbell's Values for Carbon and Carbon Plus Base.*

	Basic O. H. Steel.		Acid O. H. Steel.	
Carbon. Per Cent.	Carbon .01 = 950 Lbs.	Carbon + 37,430 Lbs.	Carbon .01 = 1,210 Lbs.	Carbon + 38,600 Lbs.
	Lbs.	Lbs.	Lbs.	Lbs.
.05	4,750	42,180	6,050	44,650
.06	5,700	43,130	7,260	45,860
.07	6,650	44,080	8,470	47,070
.08	7,600	45,030	9,680	48,280
.09	8,550	45,980	10,890	49,490
.10	9,500	46,930	12,100	50,700
.11	10,450	47,880	13,310	51,910
.12	11,400	48,830	14,520	53,120
.13	12,350	49,780	15,730	54,330
.14	13,300	50,730	16,940	55,540
.15	14,250	51,680	18,150	56,750
.16	15,200	52,630	19,360	57,960
.17	16,150	53,580	20,570	59,170
.18	17,100	54,530	21,780	60,380
.19	18,050	55,480	22,990	61,590
.20	19,000	56,430	24,200	62,800
.21	19,950	57,380	25,410	64,010
.22	20,900	58,330	26,620	65,220
.23	21,850	59,280	27,830	66,430
.24	22,800	60,230	29,040	67,640
.25	23,750	61,180	30,250	68,850
.26	24,700	62,130	31,460	70,060
.27	25,650	63,080	32,670	71,270
.28	26,600	64,030	33,880	72,480
.29	27,550	64,980	35,090	73,690
.30	28,500	65,930	36,300	74,900
.31	29,450	66,880	37,510	76,110
.32	30,400	67,830	38,720	77,320
.33	31,350	68,780	39,930	78,530
.34	32,300	69,730	41,140	79,740
.35	33,250	70,680	42,350	80,950
.36	34,200	71,630	43,560	82,160
.37	35,150	72,580	44,770	83,370
.38	36,100	73,530	45,980	84,580
.39	37,050	74,480	47,190	85,790
.40	38,000	75,430	48,400	87,000
.41	38,950	76,380	49,610	88,210
.42	39,900	77,330	50,820	89,420
.43	40,850	78,280	52,030	90,630
.44	41,800	79,230	53,240	91,840
.45	42,750	80,180	54,450	93,050
.46	43,700	81,130	55,660	94,260
.47	44,650	82,080	56,870	95,470
.48	45,600	83,030	58,080	96,680
.49	46,550	83,980	59,290	97,890
.50	47,500	84,930	60,500	99,100
.51	48,450	85,880	61,710	100,310
.52	49,400	86,830	62,920	101,520
.53	50,350	87,780	64,130	102,730
.54	51,300	88,730	65,340	103,940
.55	52,250	89,680	66,550	105,150
.56	53,200	90,630	67,760	106,360
.57	54,150	91,580	68,970	107,570
.58	55,100	92,530	70,180	108,780
.59	56,050	93,480	71,390	109,990
.60	57,000	94,430	72,600	111,200

TABLE XVI.—*Campbell's Values of Phosphorus in Acid Steel.*
(.01 *Per Cent. Phosphorus* = 890 *Lbs. per Square Inch.*)

Phosphorus. Per Cent.	.000	.001	.002	.003	.004	.005	.006	.007	.008	.009
	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.
.00..........	0	89	178	267	356	445	534	623	712	801
.01..........	890	979	1,068	1,157	1,246	1,335	1,424	1,513	1,602	1,691
.02..........	1,780	1,869	1,958	2,047	2,136	2,225	2,314	2,403	2,492	2,581
.03..........	2,670	2,759	2,848	2,937	3,026	3,115	3,204	3,293	3,382	3,471
.04	3,560	3,649	3,738	3,827	3,916	4,005	4,094	4,183	4,272	4,361
.05..........	4,450	4,539	4,628	4,717	4,806	4,895	4,984	5,073	5,162	5,251
.06	5,340	5,429	5,518	5,607	5,696	5,785	5,874	5,963	6,052	6,141
.07..........	6,230	6,319	6,408	6,497	6,586	6,675	6,764	6,853	6,942	7,031
.08..........	7,120	7,209	7,298	7,387	7,476	7,565	7,654	7,743	7,832	7,921
.09..........	8,010	8,099	8,188	8,277	8,366	8,455	8,544	8,633	8,722	8,811
.10..........	8,900	8,989	9,078	9,167	9,256	9,345	9,434	9,523	9,612	9,701

TABLE XVII.—*Campbell's Values of Phosphorus in Basic Steel.*
(.01 *Per Cent. Phosphorus* = 1,050 *Lbs. per Square Inch.*)

Phosphorus Per Cent.	000	.001	.002	003	.004	.005	.006	.007	.008	.009
	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.
.00..........	0	105	210	315	420	525	630	735	840	945
.01..........	1,050	1,155	1,260	1,365	1,470	1,575	1,680	1,785	1,890	1,995
.02..........	2,100	2,205	2,310	2,415	2,520	2,625	2,730	2,835	2,940	3,045
.03..........	3,150	3,255	3,360	3,465	3,570	3,675	3,780	3,885	3,990	4,095
.04..........	4,200	4,305	4,410	4,515	4,620	4,725	4,830	4,935	5,040	5,145
.05..........	5,250	5,355	5,460	5,565	5,670	5,775	5,880	5,985	6,090	6,195
.06..........	6,300	6,405	6,510	6,615	6,720	6,825	6,930	7,035	7,140	7,245
.07.	7,350	7,455	7,560	7,665	7,770	7,875	7,980	8,085	8,190	8,295
.08..........	8,400	8,505	8,610	8,715	8,820	8,925	9,030	9,135	9,240	9,345
.09..........	9,450	9,555	9,660	9,765	9,870	9,975	10,080	10,185	10,290	10,395
.10..........	10,500	10,605	10,710	10,815	10,920	11,025	11,130	11,235	11,340	11,445

TABLE XVIII.—*Estimated Ultimate Strengths by Campbell's Values for Acid O. H. Steel.*

Base = 38,600 lbs. per square inch for iron.
Increase of 1,210 lbs. for each .01 per cent. of carbon.
" 890 lbs. for each .01 per cent. of phosphorus.

Carbon.	.06	.07	.08	.09	.10	.11	.12	.13	.14	.15
Phos. .000.....	45,860	47,070	48,280	49,490	50,700	51,910	53,120	54,330	55,540	56,750
" .005.....	46,305	47,515	48,725	49,935	51,145	52,355	53,565	54,775	55,985	57,195
" .01	46,750	47,960	49,170	50,380	51,590	52,800	54,010	55,220	56 430	57,640
" .015.....	47,195	48,405	49,615	50,825	52,035	53,245	54,455	55,665	56,875	58,085
" .02	47,640	48,850	50,060	51,270	52,480	53,690	54,900	56,110	57,320	58,530
" .025.....	48,085	49,295	50,505	51,715	52,925	54,135	55,345	56,555	57,765	58,975
" .03	48,530	49,740	50,950	52,160	53,370	54,580	55,790	57,000	58,210	59,420
" .035.....	48,975	50,185	51,395	52,605	53,815	55,025	56,235	57,445	58,655	59,865
" .04	49,420	50,630	51,840	53 050	54,260	55,470	56,680	57,890	59,100	60,310
" .045.....	49,865	51,075	52,285	53,495	54,705	55,915	57,125	58,335	59,545	60,755
" .05	50,310	51,520	52,730	53,940	55,150	56,360	57,570	58,780	59,990	61,200
" .055.....	50,755	51,965	53,175	54,385	55,595	56,805	58,015	59,225	60,435	61,645
" .06	51,200	52,410	53,620	54,830	56,040	57,250	58,460	59,670	60,880	62,090
" .065.....	51,645	52,855	54,065	55,275	56,485	57,695	58,905	60,115	61,325	62,535
" .07	52,090	53,300	54,510	55,720	56,930	58,140	59,350	60,560	61,770	62,980
" .075. ...	52,535	53,745	54,955	56,165	57,375	58,585	59,795	61,005	62,215	63,425
" .08	52,980	54,190	55,400	56,610	57,820	59,030	60,240	61,450	62,660	63,870
" .085.....	53,425	54,635	55,845	57,055	58,265	59,475	60,685	61,895	63,105	64,315
" .09	53,870	55,080	56,290	57,500	58,710	59,920	61,130	62,340	63,550	64,760
" .095.....	54,315	55,525	56,735	57,945	59,155	60,365	61,575	62,785	63,995	65,205
" .10	54,760	55,970	57,180	58,390	59,600	60,810	62,020	63,230	64,440	65,650
.001 Phos. =	89 lbs.	89 lbs.	89 lbs.	89 lbs.	89 lbs.	89 lbs.	89 lbs.	89 lbs.	89 lbs.	89 lbs.

Carbon.	.16	.17	.18	.19	.20	.21	.22	.23	.24	.25
Phos. .000.....	57,960	59,170	60,380	61,590	62,800	64,010	65,220	66,430	67,640	68,850
" .005.....	58,405	59,615	60,825	62,035	63,245	64,455	65,665	66,875	68,085	69,295
" .01	58,850	60,060	61,270	62,480	63,690	64,900	66,110	67,320	68,530	69,740
" .015.....	59,295	60,505	61,715	62,925	64,135	65,345	66,555	67,765	68,975	70,185
" .02	59,740	60,950	62,160	63,370	64,580	65,790	67,000	68,210	69,420	70,630
" .025.....	60,185	61,395	62,605	63,815	65,025	66,235	67,445	68,655	69,865	71,075
" .03	60,630	61,840	63,050	64,260	65,470	66,680	67,890	69,100	70,310	71,520
" .035.....	61,075	62,285	63,495	64,705	65,915	67,125	68,335	69,545	70,755	71,965
" .04	61,520	62,730	63,940	65,150	66,360	67,570	68,780	69,990	71,200	72,410
" .045.....	61,965	63,175	64,385	65,595	66,805	68,015	69,225	70,435	71,645	72,855
" .05	62,410	63,620	64,830	66,040	67,250	68,460	69,670	70,880	72,090	73,300
" .055.....	62,855	64,065	65,275	66,485	67,695	68,905	70,115	71,325	72,535	73,745
" .06	63,300	64,510	65,720	66,930	68,140	69,350	70,560	71,770	72,980	74,190
" .065.....	63,745	64,955	66,165	67,375	68,585	69,795	71,005	72,215	73,425	74,635
" .07	64,190	65,400	66,610	67,820	69,030	70,240	71,450	72,660	73,870	75,080
" .075. ...	64,635	65,845	67,055	68,265	69,475	70,685	71,895	73,105	74,315	75,525
" .08	65,080	66,290	67,500	68,710	69,920	71,130	72,340	73,550	74,760	75,970
" .085.....	65,525	66,735	67,945	69,155	70,365	71,575	72,785	73,995	75,205	76,415
" .09	65,970	67,180	68,390	69,600	70,810	72,020	73,230	74,440	75,650	76,860
" .095.....	66,415	67,625	68,835	70,045	71,255	72,465	73,675	74,885	76,095	77,305
" .10	66,860	68,070	69,280	70,490	71,700	72,910	74,120	75,330	76,540	77,750
.001 Phos. =	89 lbs.	89 lbs.	89 lbs.	89 lbs.	89 lbs.	89 lbs.	89 lbs.	89 lbs.	89 lbs.	89 lbs.

TABLE XIX.—*Estimated Ultimate Strengths by Campbell's Values for Basic O. H. Steel.*

Base = 37,430 lbs. per square inch for iron.
Increase of 950 lbs. for each .01 per cent. carbon.
" 1,050 lbs. for each .01 per cent. phosphorus.

Carbon.	.06	.07	.08	.09	.10	.11	.12	.13	.14	.15	.16	.17	.18	.19	.20	.21	.22	.23	.24	.25
Phos. .000.....	43,130	44,080	45,030	45,980	46,930	47,880	48,830	49,780	50,730	51,680	52,630	53,580	54,530	55,480	56,430	57,380	58,330	59,280	60,230	61,180
" .005.....	43,655	44,605	45,555	46,505	47,455	48,405	49,355	50,305	51,255	52,205	53,155	54,105	55,055	56,005	56,955	57,905	58,855	59,805	60,755	61,705
" .01	44,180	45,130	46,080	47,030	47,980	48,930	49,880	50,830	51,780	52,730	53,680	54,630	55,580	56,530	57,480	58,430	59,380	60,330	61,280	62,230
" .015.....	44,705	45,655	46,605	47,555	48,505	49,455	50,405	51,355	52,305	53,255	54,205	55,155	56,105	57,055	58,005	58,955	59,905	60,855	61,805	62,755
" .02	45,230	46,180	47,130	48,080	49,030	49,980	50,930	51,880	52,830	53,780	54,730	55,680	56,630	57,580	58,530	59,480	60,430	61,380	62,330	63,280
" .025.....	45,755	46,705	47,655	48,605	49,555	50,505	51,455	52,405	53,355	54,305	55,255	56,205	57,155	58,105	59,055	60,005	60,955	61,905	62,855	63,805
" .03	46,280	47,230	48,180	49,130	50,080	51 030	51,980	52,930	53,880	54,830	55,780	56,730	57,680	58,630	59,580	60,530	61,480	62,430	63,380	64,330
" .035.....	46,805	47,755	48,705	49,655	50,605	51,555	52,505	53,455	54,405	55,355	56,305	57,255	58,205	59,155	60,105	61,055	62,005	62,955	63,905	64,855
" .04	47,330	48,280	49.230	50,180	51,130	52,080	53,030	53,980	54,930	55,880	56,830	57,780	58,730	59,680	60,630	61,580	62,530	63,480	64,430	65,380
" .045.....	47,855	48,805	49.755	50,705	51,655	52,605	53,555	54,505	55,455	56,405	57,355	58,305	59,255	60,205	61,155	62,105	63,055	64,005	64,955	65,905
" .05	48,380	49,330	50,280	51.230	52,180	53.130	54,080	55,030	55,980	56,930	57,880	58,830	59,780	60,730	61,680	62,630	63,580	64,530	65,480	66,430
" .045.....	48,905	49,855	50,805	51,755	52,705	53,655	54,605	55,555	56,505	57,455	58,405	59,355	60,305	61,255	62,205	63,155	64,105	65,055	66,005	66,955
" .06	49,430	50,380	51,330	52,280	53,230	54,180	55,130	56,080	57,030	57,980	58,930	59,880	60,830	61,780	62,730	63,680	64,630	65,580	66,530	67,480
" .065.....	49,955	50,905	51,855	52,805	53,755	54,705	55,655	56,605	57,555	58,505	59,455	60,405	61,355	62,305	63,255	64,205	65,155	66,105	67,055	68,005
" .07	50,480	51,430	52,380	53,330	54,280	55,230	56,180	57,130	58,080	59,030	59 980	60,930	61,880	62,830	63,780	64,730	65,680	66,630	67,580	68,530
" .075.....	51,005	51,955	52,905	53,855	54,805	55,755	56,705	57,655	58,605	59,555	60,505	61,455	62,405	63,355	64.305	65,255	66,205	67,155	68,105	69,055
" .08	51,530	52,480	53,430	54,380	55,330	56,280	57,230	58,180	59,130	60,080	61,030	61,980	62,930	63,880	64,830	65,780	66,730	67,680	68,630	69,580
" .085.....	52,055	53,005	53,955	54,905	55,855	56,805	57,755	58,705	59,655	60,605	61,555	62,505	63,455	64,405	65,355	66,305	67,255	68,205	69,155	70,105
" .09	52,580	53,530	54,480	55,430	56,380	57,330	58,280	59,230	60.180	61,130	62,080	63,030	63,980	64,930	65,880	66,830	67,780	68,730	69,680	70,630
" .095.....	53,105	54,055	55,005	55,955	56,905	57,855	58,805	59,755	60,705	61,655	62,605	63,555	64,505	65,455	66,405	67,355	68,305	69,255	70,205	71,155
" .10	53,630	54,580	55,530	56,480	57,430	58,380	59,330	60,280	61,230	62,180	63,130	64,080	65,030	65,980	66,930	67,880	68,830	69,780	70,730	71,680
.001 Phos. =..	105 lbs.	105 lbs.	105 lbs.	105 lbs.	105 lbs.	105 lbs.	105 lbs.	105 lbs.	105 lbs	105 lbs.	105 lbs.	105 lbs.	105 lbs.	105 lbs.	105 lbs.	105 lbs.	105 lbs.	105 lbs.	105 lbs.	105 lbs.

TABLE XX.—*Campbell's Values for Manganese in Basic O. H. Steel.* (.01 *Per Cent.* = 85 *Lbs. Per Square Inch.*)

Mn. Per Cent.	Lbs.	Mn. Per Cent.	Lbs.	Mn. Per Cent.	Lbs.	Mn. Per Cent.	Lbs.
		.26	2,210	.51	4,335	.76	6,460
		.265	2,252	.515	4,377	.765	6,502
		.27	2,295	.52	4,420	.77	6,545
		.275	2,337	.525	4,462	.775	6,587
		.28	2,380	.53	4,505	.78	6,630
		.285	2,422	.535	4,547	.785	6,672
		.29	2,465	.54	4,590	.79	6,715
		.295	2,507	.545	4,632	.795	6,757
		.30	2,550	.55	4,675	.80	6,800
		.305	2,592	.555	4,717	.805	6,842
		.31	2,635	.56	4,760	.81	6,885
		.315	2,677	.565	4,802	.815	6,927
		.32	2,720	.57	4,845	.82	6,970
		.325	2,762	.575	4,887	.825	7,012
		.33	2,805	.58	4,930	.83	7,055
		.335	2,847	.585	4,972	.835	7,097
		.34	2,890	.59	5,015	.84	7,140
		.345	2,932	.595	5,057	.845	7,182
		.35	2,975	.60	5,100	.85	7,225
		.355	3,017	.605	5,142	.855	7,267
		.36	3,060	.61	5,185	.86	7,310
		.365	3,102	.615	5,227	.865	7,352
		.37	3,145	.62	5,270	.87	7,395
		.375	3,187	.625	5,312	.875	7,437
		.38	3,230	.63	5,355	.88	7,480
		.385	3,272	.635	5,397	.885	7,522
		.39	3,315	.64	5,440	.89	7,565
		.395	3,357	.645	5,482	.895	7,607
.15	1,275	.40	3,400	.65	5,525	.90	7,650
.155	1,317	.405	3,442	.655	5,567	.905	7,692
.16	1,360	.41	3,485	.66	5,610	.91	7,735
.165	1,402	.415	3,527	.665	5,652	.915	7,777
.17	1,445	.42	3,570	.67	5,695	.92	7,820
.175	1,487	.425	3,612	.675	5,737	.925	7,862
.18	1,530	.43	3,655	.68	5,780	.93	7,905
.185	1,572	.435	3,697	.685	5,822	.935	7,947
.19	1,615	.44	3,740	.69	5,865	.94	7,990
.195	1,657	.445	3,782	.695	5,907	.945	8,032
.20	1,700	.45	3,825	.70	5,950	.95	8,075
.205	1,742	.455	3,867	.705	5,992	.955	8,117
.21	1,785	.46	3,910	.71	6,035	.96	8,160
.215	1,827	.465	3,952	.715	6,077	.965	8,202
22	1,870	.47	3,995	.72	6,120	.97	8,245
.225	1,912	.475	4,037	.725	6,162	.975	8,287
.23	1,955	.48	4,080	.73	6,205	.98	8,330
.235	1,997	.485	4,122	.735	6,247	.985	8,372
.24	2,040	.49	4,165	.74	6,290	.99	8,415
.245	2,082	.495	4,207	.745	6,332	.995	8,457
.25	2,125	.50	4,250	.75	6,375	1.000	8,500
.255	2,167	.505	4,292	.755	6,417		

TABLE XXI.—*Cunningham's Values for Carbon and Carbon Plus Base of* 40,000 *Lbs.*

Carbon.	Carbon Only.	Carbon + 40,000 Lbs.
Per Cent.	Lbs.	Lbs.
.05	5,000	45,000
.06	6,000	46,000
.07	7,000	47,000
.08	8,000	48,000
.09	9,000	49,000
.10	10,000	50,000
.11	11,000	51,000
.12	12,000	52,000
.13	13,000	53,000
.14	14,000	54,000
.15	15,000	55,000
.16	16,000	56,000
.17	17,000	57,000
.18	18,000	58,000
.19	19,000	59,000
.20	20,000	60,000
.21	21,000	61,000
.22	22,000	62,000
.23	23,000	63,000
.24	24,000	64,000
.25	25,000	65,000
.26	26,000	66,000
.27	27,000	67,000
.28	28,000	68,000
.29	29,000	69,000
.30	30,000	70,000
.31	31,000	71,000
.32	32,000	72,000
.33	33,000	73,000
.34	34,000	74,000
.35	35,000	75,000
.36	36,000	76,000
.37	37,000	77,000
.38	38,000	78,000
.39	39,000	79,000
.40	40,000	80,000
.41	41,000	81,000
.42	42,000	82,000
.43	43,000	83,000
.44	44,000	84,000
.45	45,000	85,000
.46	46,000	86,000
.47	47,000	87,000
.48	48,000	88,000
.49	49,000	89,000
.50	50,000	90,000
.51	51,000	91,000
.52	52,000	92,000
.53	53,000	93,000
.54	54,000	94,000
.55	55,000	95,000
.56	56,000	96,000
.57	57,000	97,000
.58	58,000	98,000
.59	59,000	99,000
.60	60,000	100,000

TABLE XXII.—*Cunningham's Values for Phosphorus.* (.01 *Per Cent. Phosphorus* = 1,000 *Lbs. Per Square Inch.*)

Phosphorus. Per Cent.	.000	.001	.002	.003	.004	.005	.006	.007	.008	.009
	Lbs.	Lbs.	Lbs.	Lbs	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.
.00...........	000	100	200	300	400	500	600	700	800	900
.01...........	1,000	1,100	1,200	1,300	1,400	1,500	1,600	1,700	1,800	1,900
.02...........	2,000	2,100	2,200	2,300	2,400	2,500	2,600	2,700	2,800	2,900
.03...........	3,000	3,100	3,200	3,300	3,400	3,500	3,600	3,700	3,800	3,900
.04...........	4,000	4,100	4,200	4,300	4,400	4,500	4,600	4,700	4,800	4,900
.05...........	5,000	5,100	5,200	5,300	5,400	5,500	5,600	5,700	5,800	5,900
.06...........	6,000	6,100	6,200	6,300	6,400	6,500	6,600	6,700	6,800	6,900
.07...........	7,000	7,100	7,200	7,300	7,400	7,500	7,600	7,700	7,800	7,900
.08...........	8,000	8,100	8,200	8,300	8,400	8,500	8,600	8,700	8,800	8,900
.09...........	9,000	9,100	9,200	9,300	9,400	9,500	9,600	9,700	9,800	9,900
.10...........	10,000	10,100	10,200	10,300	10,400	10,500	10,600	10,700	10,800	10,900

TABLE XXIII.—*Estimated Ultimate Strengths by Cunningham's Values for Acid and Basic O. H. Steels.*

Base = 40,000 lbs. per square inch for iron.
Increase of 1,000 lbs. for each .01 per cent. of phosphorus.
" 1,000 lbs. for each .01 per cent. of carbon.

Carbon.	.06	.07	.08	.09	.10	.11	.12	.13	.14	.15	.16	.17	.18	.19	.20	.21	22	.23	.24	.25
Phos. .000	46,000	47,000	48,000	49,000	50,000	51,000	52,000	53,000	54 000	55,000	56,000	57,000	58,000	59,000	60,000	61,000	62,000	63,000	64,000	65,000
" .005	46,500	47,500	48,500	49 500	50,500	51,500	52,500	53,500	54,500	55,500	56,500	57,500	58,500	59,500	60,500	61,500	62,500	63,500	64,500	65,500
" .01	47,000	48,000	49,000	50,000	51,000	52,000	53,000	54,000	55,000	56,000	57,000	58,000	59,000	60,000	61,000	62,000	63,000	64,000	65,000	66,000
" .015	47,500	48,500	49,500	50,500	51,500	52,500	53,500	54,500	55,500	56,500	57,500	58,500	59,500	60,500	61,500	62,500	63,500	64,500	65,500	66,500
" .02	48,000	49,000	50,000	51,000	52,000	53,000	54,000	55,000	56,000	57,000	58,000	59,000	60,000	61,000	62,000	63,000	64,000	65,000	66,000	67,000
" .025	48,500	49,500	50,500	51,500	52,500	53,500	54,500	55,500	56,500	57,500	58,500	59,500	60,500	61,500	62,500	63,500	64,500	65,500	66,500	67,500
" .03	49,000	50,000	51,000	52,000	53,000	54,000	55,000	56,000	57,000	58,000	59,000	60,000	61,000	62,000	63,000	64,000	65,000	66,000	67,000	68,000
" .035	49,500	50,500	51,500	52,500	53,500	54,500	55,500	56,500	57,500	58,500	59,500	60,500	61,500	62,500	63,500	64,500	65,500	66,500	67,500	68,500
" .04	50,000	51,000	52,000	53,000	54,000	55,000	56,000	57,000	58,000	59,000	60,000	61,000	62,000	63,000	64,000	65,000	66,000	67,000	68,000	69,000
" .045	50,500	51,500	52,500	53,500	54,500	55,500	56,500	57,500	58,500	59,500	60,500	61,500	62,500	63,500	64,500	65,500	66,500	67,500	68,500	69,500
" .05	51,000	52,000	53,000	54,000	55,000	56,000	57,000	58,000	59,000	60,000	61,000	62,000	63,000	64,000	65,000	66,000	67,000	68,000	69,000	70,000
" .055	51,500	52,500	53,500	54,500	55,500	56,500	57,500	58,500	59,500	60,500	61,500	62,500	63,500	64,500	65,500	66,500	67,500	68,500	69,500	70,500
" .06	52,000	53,000	54,000	55,000	56,000	57,000	58,000	59,000	60,000	61,000	62,000	63,000	64,000	65,000	66,000	67,000	68,000	69,000	70,000	71,000
" .065	52,500	53,500	54,500	55,500	56,500	57,500	58,500	59,500	60,500	61,500	62,500	63,500	64,500	65,500	66,500	67,500	68,500	69,500	70,500	71,500
" .07	53,000	54,000	55,000	56,000	57,000	58,000	59,000	60,000	61,000	62,000	63,000	64,000	65,000	66,000	67,000	68,000	69,000	70,000	71,000	72,000
" .075	53,500	54,500	55,500	56,500	57,500	58,500	59,500	60,500	61,500	62,500	63.500	64,500	65,500	66,500	67,500	68.500	69,500	70,500	71,500	72,500
" .08	54,000	55,000	56,000	57,000	58,000	59,000	60 000	61,000	62,000	63,000	64,000	65,000	66,000	67,000	68,000	69.000	70,000	71,000	72,000	73,000
" .085	54,500	55,500	56,500	57,500	58,500	59.500	60,500	61,500	62,500	63,500	64,500	65,500	66,500	67,500	68,500	69,500	70,500	71,500	72,500	73,500
" .09	55,000	56,000	57,000	58,000	59,000	60,000	61,000	62,000	63,000	64,000	65,000	66,000	67,000	68,000	69,000	70,000	71,000	72,000	73,000	74,000
" .095	55,500	56,500	57,500	58.500	59,500	60,500	61,500	62,500	63,500	64,500	65,500	66,500	67,500	68,500	69,500	70,500	71,500	72,500	73,500	74,500
" .10	56,000	57,000	58,000	59,000	60,000	61,000	62,000	63,000	64,000	65,000	66,000	67,000	68,000	69,000	70,000	71,000	72,000	73,000	74,000	75,000
.001 Phos. =	100 lbs.	100 lbs.	100 lbs.	100 lbs.	100 lbs.	100 lbs.	100 lbs.	100 lbs	100 lbs.	100 lbs.	100 lbs.	100 lbs.	100 lbs.	100 lbs.	100 lbs.	100 lbs.	100 lbs.	100 lbs.	100 lbs.	100 lbs.

Notes on Tuyeres in the Iron Blast-Furnace.

A Contribution to the Discussion on this Subject.

BY JOHN M. HARTMAN, PHILADELPHIA, PA.

(Buffalo Meeting, October, 1898.)

An examination as to irregularity of wear around the nose of the Witherbee tuyeres showed a section through the nose near the top as per Fig. 1, and a section only a half-inch beyond as per Fig. 2. The molten iron flowing over the nose of the tuyere had melted off the metal until the water in the tuyere prevented further melting.

These tuyeres have a division in the annular water-space (see Fig. 2) to turn the current of water at the top, and the water-supply pipe is carried to within 1 inch of the nose. The constant supply of cold water in the nose of the tuyere kept it well cooled and preserved. But half an inch on the other side of the division the metal had melted off, as is shown by Fig. 1. There was no means of effecting a circulation at this point so long as the water was discharged from the butt of the tuyere. A pipe having been placed on the discharge-side of the division, the same as on the supply-side, in order to secure a circulation of the water at the nose on the discharge-side, it was found that the metal did not burn off. The nose being found to give out at the bottom, a division was placed in the annular water-space at the bottom, terminating 1 inch from the nose. This made a sharp current of circulation in the nose at the bottom, and prevented the burning of the metal there.

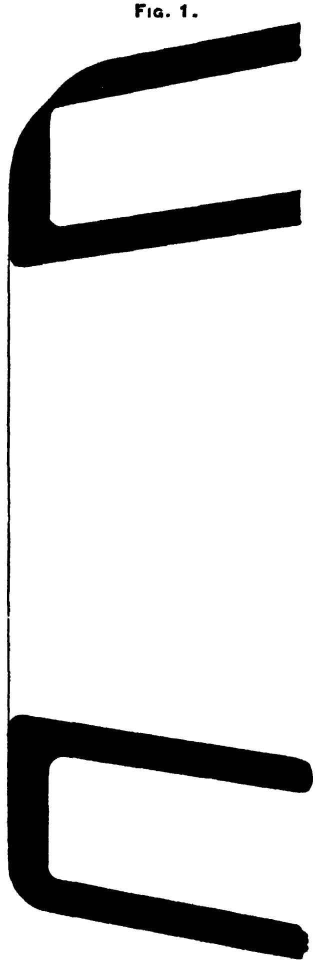
A positive circulation at the nose, top and bottom, is a necessity with the present fast driving of furnaces, and prolongs the life of the tuyeres.

The proper metallic composition of tuyeres is a question constantly agitated. A trial of ordinary yellow-metal* tuyeres was made 12 years ago. They did their work just as well

* An alloy of copper and zinc, with a pretty high percentage of zinc.

as the 97 per cent. copper tuyeres alongside of them, and presented the decided advantage that they did not crack when the furnace slipped and pushed the molten iron up around the tuyeres, or showers of iron came from above, causing them to get red-hot. In May of this year a 6-inch yellow-metal tuyere was placed in a furnace using sulphurous ores and receiving about 25,000 feet of air per minute. A letter from the owners of this furnace, dated October 13, 1898, says that the tuyere is still in use.*

FIG. 1.



Witherbee Bronze Tuyere. Nearly Central Vertical Section through Nose, Showing Effect of Molten Iron on Top.

When melted iron running over a tuyere, or pushed up against it, makes it red-hot, a bubble of steam is formed on the hot interior surface, increasing in size and pressure until it overcomes the water-pressure, when the bubble bursts and the water rushes against the hot surface; but, the metal having become heated, a second and larger bubble forms and explodes, blowing the water out of the tuyere. With a good water-circulation the iron of an iron tuyere is so chilled as to resist this sudden change; and yellow metal does not crack, though the tuyere becomes red-hot, but remains as tough as when cold. Copper tuyeres, under the same circumstances, are liable to crack. Pure copper tuyeres, moreover, are too soft, are worn off by the abrasion of the stock, spring out of shape if they are "dollied" in the furnace, and crack when subjected to a high heat.

As for the higher conducting-power of pure copper tuyeres,

* In January, 1899, the tuyere was still at work.

it has not yet been proven in practice. Pure copper castings are likely to be more or less spongy, as can be demonstrated by etching with acid a freshly-broken surface. A small percentage of zinc, added to the copper, makes a more solid tuyere-casting, which withstands abrasion, and conducts heat practically as well as pure copper.

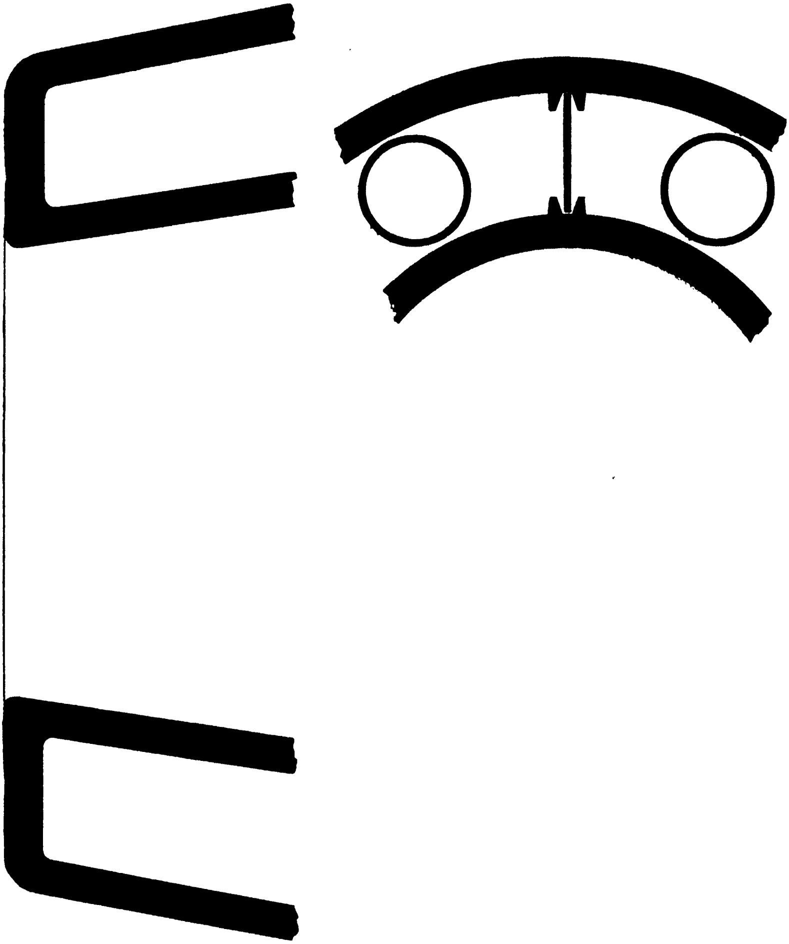
A difficulty with which tuyere-makers have to contend is the return by furnace-managers of tuyeres that are cracked and the demand for new ones to replace them, on the assumption that they were originally defective. This cracking is caused by the formation, above the tuyeres, of pockets holding molten iron. When the blast is thrown off the furnace the stock settles, and, pushing down, breaks off these pockets, letting the molten iron shower down over the nose of the tuyere. This occurs only when the furnace is working irregularly. It is evidenced by the grooves cut on the tuyere-nose, which weaken the metal and eventually cause a crack. With a good positive water-circulation at the nose, tuyeres stand this shower of iron much better. The grooves cut on the top of the nose by the showers of iron show that such pockets have existed. When a furnace not containing such pockets slips and pushes the molten iron in the hearth up around the tuyere-nose, thus, perhaps, burning it, there are no grooves; the action lasts only about two seconds, and is uniform over the whole surface. The descending showers of molten iron upon the nose are more serious causes of injury than the pushing-up of iron around the tuyeres.

Another trouble may arise from decreasing the volume of air and not decreasing correspondingly the area of the nozzles. When a furnace shows by its behavior that it is receiving too much air the blast is decreased, and, the nozzles being then too large, the blast reacts on the wall over the tuyere, melting away the self-protecting coating, and, working hard on the walls, melts iron and slag immediately above the tuyere, which run down over the tuyere-nose and soon destroy it. Why did not the founder reduce his nozzles at once, get more penetration, and avoid this trouble? The laws of the blast-furnace are rigid; and the change of one factor in its operation demands a rational regard for the other factors thereby affected.

When a furnace is running regularly, the water in the tuyere chills the cinder and shots of iron around the nose of the tuyere,

forming a self-protecting coating. This coating is specially valuable to a new tuyere, just put in, and such a tuyere should be left for an hour before air is put through it, in order to give it a chance to build up.

FIG. 2.



Witherbee Bronze Tuyere. Vertical Section ½ inch beyond Fig. 1, and Cross-Section Showing Water-Circulation.

The proper penetration into the crucible of the air from the nose of the tuyere is a question not decided; but, as larger hearths are used, more penetration is required. This is, of course, a function of the diameter of the nozzle. Some years ago 5-inch tuyeres as a maximum for anthracite and 6-inch for coke-furnaces were fixed by the writer, and up to this date no

reasons have been found for a change. A minimum velocity of 20,000 feet a minute for anthracite and 15,000 feet for coke-furnaces works well for round-nose tuyeres in crucibles up to 10-foot diameter.

Penetration is also modified by the heat in the crucible. The hotter the fuel in the crucible, the greater is the avidity of air for the fuel, and the farther in it will go. As the fuel decreases in heat, the pressure goes up; the loss by leaks increases; the furnace receives less air and less heat is generated; but the constant loss from radiation, tuyere-water, jackets and bosh-plates continues; so that a furnace soon goes from bad to worse unless measures are at once taken to get the crucible hot.

The air from the tuyeres, coming in contact with the fuel, reduces it to minute particles; but this result is affected by changes in the current of the air. A piece of fuel, falling before the tuyere-nose, disperses the blast sidewise and up-and-down, creating a hole which permits the fuel to drop down and form a dam to disperse the blast. This may continue until a hole is formed above, which allows the air to go up along the walls at that point, causing irregular work. Perhaps, at the next tuyere, the air is penetrating properly through a well-defined opening in the fuel; but presently a lump of fuel falls down from the top of this opening, and the blast is dispersed sidewise. At the next tuyere the fuel may be still settling steadily and evenly; so this problem of securing proper penetration is an intricate one.

Currents of gas from the combustion of the fuel, sweeping up through the bosh, should keep the fuel clean from dust; and this dust is normally caught in the zone of carbonization, just above the zone of fusion. When dust gets down into the crucible it may smother the fuel and cause cold working.

When one of the tuyeres gets more than its share of air the current works up the wall, burns a hole through the zone of fusion, and lets the disintegrated fuel, ore and stone fall from the zone of carbonization down into the crucible. The analyses of the dust or fine stuff thus dropping below its proper position in the furnace show it to be the disintegrated stock. The resulting trouble is generally called a dust-scaffold; but this is a misnomer. It is simply an indication that irregular air-dis-

tribution has worked a hole up into the zone of carbonization. This zone is an important one, since it determines the grade of the iron produced. It may get too thick, and cause high pressure; but it will only thicken with cold working and slow driving, causing the stock to disintegrate faster than it is melted.

Such dust in front of the tuyeres prevents the air from reaching the center of the crucible, and hence the crucible becomes colder, and melting ceases. In one case known to the writer the descent of this material ran up the blast-pressure so far as to stop the engines; and, on shoveling out, it was found that the material had fused and chilled around the nose of each tuyere, forming a large bladder, of about 20 inches diameter.

Constant vigilance is required at a furnace to secure good results. How many furnace-men have noted the exceptionally large loss of tuyeres in the fall, when dead leaves fill the water-courses, or after a heavy freshet, when mud fills the tuyeres, or in early summer, after eels have grown large enough to choke a tuyere-pipe? Does the man manipulating a Bessemer converter turn on the air and let it run itself? Is he not watching everything with all his brain power and meeting every tendency to go wrong? Yet the manipulation of the converter is certainly not more important than that of the blast-furnace, which is really far more intricate, but is too often allowed to go as it pleases, misfortunes being charged to bad luck, or, in some cases, to bad tuyeres.

Multiple tuyeres now being on trial, the use of oval-nose tuyeres and the results got by them will be instructive.

When a better distribution of air across the crucible was desired, tuyeres were tried with the nozzle decreased in height and made wider horizontally, care being taken to keep more than full area up to the nose, to give full penetration. The first tuyere worked well for some time: but one day the cinder broke out through the joints of the tuyere-jacket, about 21 inches each way from the center of the tuyere. On examining inside the breast, it was found that the air had reacted and burned the brick-work around the breast. No cause for this effect could be clearly recognized; but the tuyere was replaced by a round-nose. On coupling-up the tuyere-pipe it was found that the "back-gas," from stoppages after successive castings,

had built up ashes in the tuyere-pipe, leaving a 3-inch hole to operate a tuyere equivalent to a 5-inch opening. This weakening of the air-current had caused the air to react on the brick-work of the breast and destroy it. Another tuyere, on the opposite side of the furnace, ran along nicely for 8 months, when the furnace became scaffolded, and a small round-nose tuyere was substituted to get penetration. The furnace went out of blast shortly after, and it was found that the brick-work was cut out more around the oval-nose tuyeres than around the round ones. These oval tuyeres were tried on 6 or 7 furnaces, and all did well until the furnaces got into trouble, when they had to take the blame for the bad work and go. The oval-nose tuyeres, when in operation with full volume of air, showed a good mellow tuyere-opening; and, on running in the pricker-rod, the fuel was found to be soft to the center. The side-dispersion of the air could be seen by noting the particles of solid matter passing through the tuyere-pipe and flying off sidewise. Ample dispersion was found to be secured with the proportions of heights, widths, and flare used.

Tuyeres have been tried with side-openings from the central opening; but they quickly destroyed the breast-walls, and showed too much dispersion.

It is a well-known fact that in the use of multiple tuyeres when the air-pressure goes up, and there is not sufficient penetration, some of the tuyeres are shut off to get more penetration. After a careful review of the experience with oval-nose tuyeres, it is found that they work best with the air passing at 20,000 feet per minute for coke- and 25,000 feet for anthracite-furnaces; and this velocity must be maintained to prevent reaction on the breast-walls as now built; but with better water-cooling these walls can be maintained. When a furnace gets into trouble, and requires more penetration, the quickest remedy is to replace the tuyere with a smaller one.

The oval tuyeres heretofore used (5-inch nose) have been applied by simply substituting the oval for round-nose tuyeres, using the same volume of air. The larger the tuyere of any kind, the greater is the penetration of the air into the crucible; and in changing from round to oval tuyeres the tuyeres should be proportioned to use double the volume of air used by any single multiple tuyere, there being but one tuyere and pipe in

place of two multiples; but the oval tuyeres should not exceed the equivalent of a 6-inch tuyere for anthracite and 7-inch for coke, each tuyere on coke supplying 12 square feet of tuyere-circle.

Another matter connected with penetration is the loss of air in leakage, which decreases the volume and the penetration. Little attention is paid to this matter, which is far more serious with fire-brick stoves than is generally recognized.

The advantages of oval-nose tuyeres consist in the better dispersion of the air over the whole tuyere-section of the crucible, the avoidance of multiple tuyeres, fewer water-connections involving possible leakage into the furnace, and fewer tuyere-breasts, tuyeres and pipes. The time is coming when, with constant care and watchfulness, the furnace will be blown with a uniform volume of air, and the oval tuyere will then have its day.

Tuyeres in the Iron Blast-Furnace.

A Contribution to the Discussion on this Subject.

BY B. F. FACKENTHAL, JR., RIEGELSVILLE, PA.

(Buffalo Meeting, October, 1898.)

THE earliest history shows that, in the reduction of iron-ores, natural draft was used to supply the blast, and that, when artificial blast was first used, it was supplied by leather bellows, usually made of skins, which were worked by the feet or hands, the blast entering the furnace through one tuyere or nozzle. The prehistoric bellows and its tuyere or nozzle are still in use in some form wherever iron is made.

The original Durham furnace, built in 1727 at Durham, Pa., which was one of the earliest blast-furnaces in this country, was blown with a bellows, the power being supplied by a water-wheel. The old records show the purchase of leather at various times to repair the bellows, and mention the freezing of the water-wheel as one of the causes for going out of blast.

In the early days of our iron manufacture but one tuyere was used; in fact, a considerable number of charcoal-furnaces continued to use but one tuyere down to recent years. I am

told that a cold-blast charcoal-furnace, situated in Berks county, Pa., built in 1792, continued to use but one tuyere as late as 1889, when the furnace was remodeled and equipped with four tuyeres.

My earliest recollection of any blast-furnace is that of the two old anthracite-furnaces at Durham. These were demolished in 1874 to make way for the one large furnace, which was first put in blast February 21, 1876. At the time I speak of these two old furnaces were blown through three tuyeres, which were made of 1-inch wrought-iron coils covered with cast-iron. The holes in the goose-necks, for pricking the tuyeres, were closed by means of a plug, with a handle about 12 inches long, made of gas-pipe. My idea at that time was that these plugs were for the sole purpose of being taken out by Uncle Billy Martin, the founder, to frighten visitors, particularly small boys, who had no special business around the furnace. In later years I learned that Uncle Billy used to determine the temperature of the blast by spitting on the tuyere-pipe. This, however, was not quite as crude as the method of the old charcoal-founder, of whom it was said that he could determine the conditions of his furnace by sticking his hand in the tail-race.

During the early blasts of the Durham furnaces, which were built in 1848–50, and were therefore among the earlier anthracite blast-furnaces, home-made wrought-iron tuyeres were used. They were known as welded tuyeres. One of these old tuyeres is, at the present time, suspended back of the furnace, where it was long used as a gong to call the fillers to the cast-house. This tuyere is a hollow truncated cone, 24 inches long; the sheet-iron rims are ¼-inch thick. The large end has 12-inch outside and 7-inch inside diameter, and the ring welded in this large end is ½-inch thick by 2 inches wide. The small or nozzle-end has 9-inch outside and 5-inch inside diameter, and the ring welded in this small end is ½-inch thick by 1½ inches wide. These tuyeres were made of sufficient length to permit them to be cut off when burnt and a new end welded in. A skillful mechanic, according to the standard of those days, was required to make these tuyeres and keep them in repair; and he usually commanded good wages.

The water-tuyere was invented and first used by Mr. Condie.

It was an invention found necessary when the hot-blast, patented by Mr. Neilson in 1828, was first applied to the blast-furnace.

At first, tuyeres made of wrought-iron coils were used; then tuyeres made of cast-iron, which were discarded for welded tuyeres, such as I have described above; but wrought-iron coil-tuyeres came again into general use, and held their own for many years, until they were gradually replaced by bronze tuyeres. I am told that cast-iron tuyeres were formerly in common use in this country. I have never seen one of them; but they are described as being square on the outside, and measuring about 18 inches at the large and 16 inches at the small end. The opening through the tuyere was circular, having about 9 inches diameter at the large end and 5 inches at the nozzle; and there were four square openings, one at each corner, for attaching the water-supply and discharge-pipes. These holes were closed with wooden plugs, in which a hole was bored and the pipe was secured with wooden wedges. Only two of these connections were used at a time. The cast metal was ¾-inch thick.

The new Durham furnace, first blown in February 21, 1876, was equipped with a Lurmann closed front, and was one of the first blast-furnaces in this country to use this arrangement successfully. The royalty paid for the right to use the patent was $1,900.00, being $100 per foot of bosh-diameter.

During the first and second blasts of this furnace but five tuyeres were used. The hearth- and bosh-walls were thick, the hearth was small, and the results obtained were not entirely satisfactory as to either output or fuel-economy; and, moreover, the furnace showed a constant tendency to scaffold. The hearth was then enlarged, and the number of tuyeres was increased to seven. These changes produced a marked improvement in every respect; but it cannot be claimed that this was due entirely to the increased number of tuyeres, though they were certainly of great benefit.

During 1876 bronze tuyeres, furnished by Mr. John M. Hartman, were experimented with at Durham for a long time. These were among the very first bronze tuyeres used in this country. They were shaped very much like the old-fashioned welded tuyeres above described, differing, however, in two es-

sential features, viz., the water-feed pipes were extended inside of the tuyere, and delivered the cold water well up to the nozzle; and there was a ground ball-joint between the belly-pipe and tuyere. Clay packing was used between tuyere and water-cooler. It was found that these bronze tuyeres gave much better service than the ordinary coil-tuyere, but their cost (about $90 each) was too great to justify their adoption. Several years later, the Witherbee short bronze tuyeres were first introduced in practice.

One of the latest improvements in blast-furnace construction is the use of a much larger number of tuyeres, some furnaces having as many as twenty. The idea of using a larger number of tuyeres, however, is not new, as the experiment was tried by Mr. Samuel Thomas at the Thomas Iron Co.'s plant, more than thirty-six years ago.

In Percy's *Metallurgy* (page 380), reference is made to the Thomas Iron Co.'s furnaces, and the drawing in the back part of that book shows No. 4 furnace, equipped with eleven tuyeres. These, however, as the drawing shows, were not spaced uniformly around the hearth. This was a large number of tuyeres to use in those early days. The operation of the furnace was successful, and the results obtained were satisfactory.

It is, however, of the Thomas Iron Co.'s No. 3 furnace, built at the same time as No. 4, that I wish especially to speak. This furnace was first put in blast July 18, 1862, and was equipped with twenty-one tuyeres, evenly spaced around the furnace. The furnace was constructed with twenty-one cast-iron columns or buck-stays, and a tuyere was placed between each two. It is to be regretted that the drawings of the tuyere and its arrangement have been lost or mislaid. I am, however, informed by Mr. Thomas, who designed the arrangement, and whose recollection is quite clear, that the tuyeres were made of 1-inch wrought-iron coils, covered with cast-iron, of a rectangular shape, with openings at the nozzle of 1½ by 9 inches. As the hearth of the furnace was 8 feet in diameter, while the tuyeres projected 3 inches into the furnace, and the cast-iron on each side of the tuyeres was at least 2 inches thick, making the outside measurement 13 inches, it is apparent that the tuyeres formed a continuous circle around the furnace. This continuous belt of blast cut away the brick-work, and caused

the tuyeres to sag and direct the current of blast towards the bottom of the hearth. This difficulty was overcome by casting, on the tops of the tuyeres, on the butt-end, lugs which rested against a cast-iron plate running continuously around the furnace, above the tuyeres. No water-coolers were used in connection with these tuyeres. Moreover, it is probable that the quantity of air used was insufficient for so large a tuyere-area. These rectangular tuyeres were replaced from time to time with ordinary coil-tuyeres, having a circular opening; but the entire number of twenty-one tuyeres, part rectangular and part round, continued in use until the end of the blast, July 10, 1863, when the furnace was forced to go out of blast for lack of men, because so many had enlisted for the war.*

I am also informed by Mr. Samuel Thomas that, in his experiment with so large a number of tuyeres, he realized the importance of having some additional cooling-arrangement, which led to the use of bosh-coils. He put in and used five courses of coils made of gas-pipe of 1½-inch internal diameter, placing them horizontally around the furnace above every fourth course of brick. This would bring them about 18 inches apart, and carry them about 7½ feet above the center-line of the tuyeres. They were built in the brick-work, with about 16 inches of brick on the outside. These bosh-coils were in use several months; but the founders were prejudiced against them and accused them of being the cause of every little trouble or irregularity in the operation of the furnace, and for that reason they were abandoned.

It can be said, however, that Mr. Thomas was the pioneer in using not only a large number of tuyeres, but the bosh-coil as well. A patent has recently been granted for the very arrangement of tuyeres which Mr Thomas used at Hokendauqua in July, 1862, more than thirty-six years ago; and bosh-coils have also since been patented.

Mr. Thomas does not claim to have used this large number

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* I find the following resolutions on the minute-book of the Company, under date of June 29, 1863:

Resolved, That the action of the President in offering to continue the pay of the men who would volunteer for the defence of the State under the call of the Governor of the State for ninety days, be approved, and that the officers of the Company are hereby authorized to carry out the arrangement.

Resolved, That No. 3 furnace be blown out.

of tuyeres successfully, satisfactory results having been prevented by imperfect cooling-arrangements; but in view of the fact that many of the latest and best-constructed furnaces are now using a large number of tuyeres, his experience adds a very valuable contribution to the literature of this subject.

During 1893–94 No. 1 furnace at Hokendauqua was rebuilt, with 17-foot bosh-diameter and 80-foot height, and eight tuyeres. It was put in blast August 1, 1894. The results were, in the main, not very satisfactory. The old iron-pipe stoves employed during the blast were too small for so large a furnace, and proved a constant hindrance to economical work.

During the fall of 1897 the old iron-pipe stoves were replaced with three fire-brick stoves, each 20 feet by 80 feet; the bosh of the furnace was enlarged to 17 feet 9 inches in diameter, and the tuyeres were increased in number from eight to twelve. The second blast was commenced January 8, 1898; and during the first six months, running on a foundry mixture, the fuel-consumption, including the coal used to blow in with, was 1 ton, 1 cwt., 2 qrs., 10 lbs. (1.08 tons) per ton of pig-iron produced. The fuel-consumption for some months was less than 1 ton per ton of iron, with a maximum output of 1327 tons of pig-iron per week. The total output for the first six months of the blast was 46 per cent. greater than for the corresponding time of the first blast. During the second blast, now in progress, the furnace, with the additional number of tuyeres, works with regularity, has but little tendency to scaffold, and, in this respect, shows a great improvement over the first blast. It is almost impossible to make a comparison of the blast-pressure between the two campaigns, as the area through the ovens during the first blast was not sufficiently large, and in consequence the back-pressure was very great. I am inclined to think, however, that the blast-pressure under the same conditions would not be less with twelve than with eight tuyeres; but in so large a furnace I am convinced that more uniform results can be obtained with the increased number of tuyeres.

The Evolution of Mine-Surveying Instruments.

BY DUNBAR D. SCOTT, IRONWOOD, MICH.

(Buffalo Meeting, October, 1898.)

THE development in the perfection of mine-surveying instruments has been by no means rapid, as it has depended somewhat on the details of construction borrowed from astronomical and geodetic theodolites, largely on the restrictions laid down by mining companies and the prejudices of mine managers themselves, but more than all on the methods used in conducting surveys and the importance attached thereto.

Mine-surveying, in some form or other, has been practiced from the very earliest times; but it has never kept pace with the other branches of surveying, or even with the art of mining itself, and cannot be recognized as an exact science until shortly before the beginning of this century.

The works of Hero of Alexandria, who lived in the second century B.C., are still extant, and contain descriptions of a rectangular sighting-instrument, which he invented and called a *diopter*. His improvement upon this simple construction, which possibly he devised for use in the Greek mines for rough leveling-purposes and for laying out any angle, must be considered, says Hübner,* as the origin of the highly perfected theodolite of to-day.

Whether this instrument came into general use during the first centuries of the Christian era, is not recorded; in fact, no writer undertakes to tell how mine-surveying was conducted until 1556, in which year Agricola expounds the principles of mining and metallurgy in his *De Re Metallica*, devoting the entire fifth chapter to the practice of mine-surveying (see Fig. 1).

In mediæval times those who possessed any knowledge of engineering skill made strenuous effort to keep their art a secret, partly on account of the miners' proverbial conservatism and partly for their own personal aggrandizement, and were, in

* *Mittheilungen aus dem Markscheidewesen*, von Werneke, Freiberg, 1887.

consequence, superstitiously regarded as sorcerers of the kind who were expert in the use of the divining-rod.

Indeed the superstition of this period was so potent in its in-

FIG. 1.

Facsimile from *De Re Metallica*, Georgius Agricola, Basel, 1556, constituting the frontispiece of Bennett H. Brough's "Treatise on Mine-Surveying," London, 1888.

fluences that the hazel-twig, in the hands of a sensitive medium, was accepted at that time with greater confidence than the most scientific mathematical deduction then possible.

Dr. Raymond published, in 1883, a very complete and interesting paper on "The Divining-Rod,* in which reference is made to all the best works on the subject, both ancient and modern. In the study of the history of the subject, he says:

"It will appear that divining-rods were first used in antiquity mainly or wholly for moral purposes; that in the Middle Ages their employment was for a long period confined to the discovery of material objects; that towards the end of the seventeenth century the moral use was again asserted, and that in the eighteenth century the divining-rod was relegated to the material sphere and assumed the comparative modest functions in the discharge of which it still lingers among us."†

And after showing that the rod itself serves, at most, to exhibit the results of nervous sensibility and unconscious muscular contraction on the part of the operator, he adds:‡

"To this, then, the rod of Moses, of Jacob, of Mercury, of Circe, of Valentine, of Beausoleil, of Vallemont, of Aymar, of Bleton, of Pennet, of Competti—even of Mr. Latimer—has come at last. In itself it is nothing. Its claims to virtues derived from Deity, from Satan, from sympathies and affinities, from corpuscular effluvia, from electrical currents, from passive perturbatory qualities of organo-electric force, are hopelessly collapsed and discarded. A whole library of learned rubbish about it, which remains to us, furnishes jargon for charlatans, marvellous tales for fools and amusement for antiquarians; otherwise it is only fit to constitute part of Mr. Caxton's *History of Human Error*. And the sphere of the divining-rod has shrunk with its authority. In one department after another it has been found useless. Even in the one application left to it with any show of reason, it is nothing unless held in skilful hands; and whoever has the skill may dispense with the rod."

Agricola says the subject is open to much dispute; states the evidence on both sides briefly, but with admirable clearness; and, while he declines to enter upon a discussion, "neither permissible nor agreeable," of the virtue which may be imparted to the rod by spells and incantations, he inclines his reader to skepticism. In the quaint wood-cut accompanying this chapter, his "good and sober" miners, who have studied nature, are already digging ore, while the man with the rod is yet preparing to discover it.

Fig. 2 is taken from the *Cosmographia Universalis* of Sebastian Munster, published at Basel in 1550. "This geographical work," says Dr. H. R. Mill, "deals only vaguely with mining,

* *Trans.*, xi., 411. † *Ibid.*, p. 413. ‡ *Ibid.*, p. 445.

and I fancy the cut of the *virgula divina*, to which little reference is made, must have been copied from some earlier work."

The Latin treatise on mining engineering by J. F. Weidler (Wittenberg, 1726) deals at length with this supernatural method; and even so clever an engineer as Beyern let superstition get the best of his mathematics. As late as 1749 he claims that thorough instruction in mining engineering involves the application of the divining-rod, though he was intelligent enough to insist that, "if there is a difference in the findings of the twig and compass, then more dependence must be placed in the compass than in the twig."

FIG. 2.

Ancient Representation of Divining-Rod.

It is recorded in Chinese annals that in 2364 B.C. the Emperor Hou-ang-ti, or Hong-Ti, constructed an instrument for indicating the South, which, Dr. Gilbert says,* was brought from Cathay to Italy in 1295 by the renowned Marco Polo.†

Flavio Gioja, of Amalfi, some ten or fifteen years later, was doubtless the first European to mount this magnetic needle in a box, but the use of the stationary or *Setz-compass* (Fig. 3) in mine-surveys is first described in the work of Agricola.

An instrument of this original type, bearing the date 1541, is still preserved at the Neudorfer mines in the Harz. Concerning it, Prof. Brathuhn says:

"The 5.5 cm. compass-box fits into the center of a wooden disk 16 cm. in diameter and 2 cm. thick. About it are three concentric grooves filled with wax of

* *Colcestrencis . . . de Magnete* Gulielmi Gilberti, London, 1600.

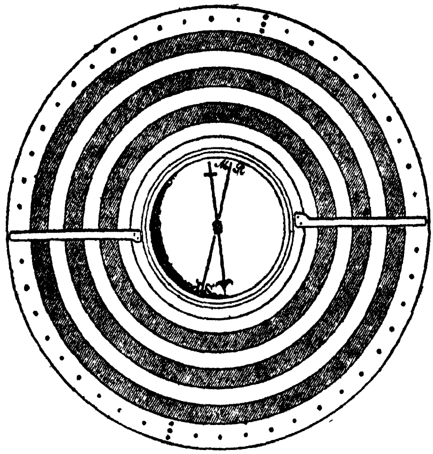
† Bailly, *Histoire de l'Astronomie Ancienne*, Paris, 1775, p. 122.

different colors. Upon the bottom plate of the compass-box is drawn only a meridian line, marked at its ends M. R. (Meridies) and S. P. (Septentrio). When in use, the compass and disk were put into the circular cavity of a wooden box, and mounted by means of a hole beneath upon a simple staff.

"The disk was turned until the needle became coincident with the meridian line; then the pointer that revolved about its fiducial edge was brought into the direction of any course, as nearly as could be judged by the eye, and a mark was made in one of the wax circles to indicate its azimuth with the meridian.

"The course was then measured and recorded with the characterized mark and the color of the wax circle in which it was made. The survey was then reproduced on the surface, commencing usually at the mouth of the shaft, to determine the proximity of the underground workings to the boundary-lines."

FIG. 3.



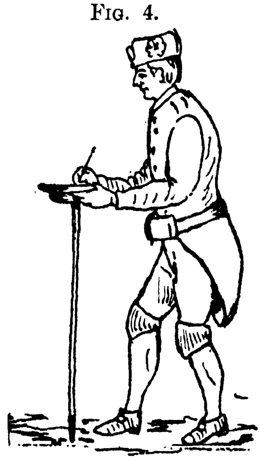
The Setz-Compass.

Fig. 4 is copied from a drawing of that period, and represents an authorized engineer, commissioned by the government of Saxony, engaged in conducting a survey with this instrument.

In another place in Agricola's work is represented a nude surveyor making observations with a circle of wood, nearly equal in diameter to his own height, which he holds vertically, and which is provided with a weighted index-pendulum.

In these two crude yet ingenious appliances we have, no doubt, the origin of the *Hangcompass und Gradbogen* that came so universally into use throughout the mining districts of Europe.

FIG. 4.

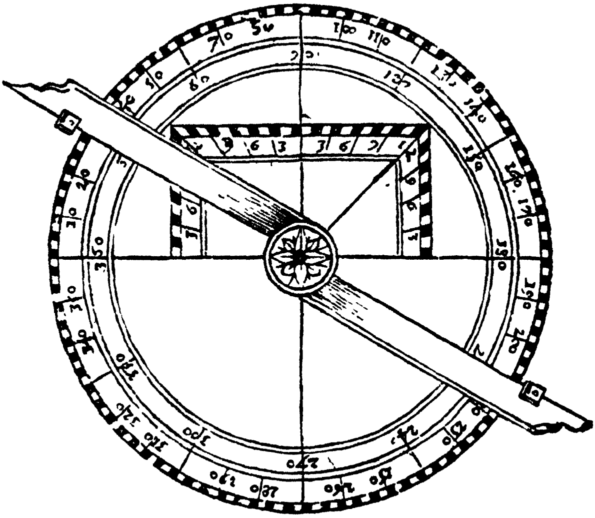


Surveying with the Setz-Compass.

In 1571 Thos. Diggs, the son of Leonhard Diggs, published in England his *Pantometria*, in which are described several instruments for surveying purposes. His masterpiece is

what he called the *theodolitus* (Fig. 5), perhaps derived from *theodicæa*, taken in the sense of perfection, as being a most per-

FIG. 5.



Diggs's Theodolite.

fect instrument.* In the 27th chapter, called *Longimetria*, he says:

"It is but a circle divided into 360 grades or degrees, or a semicircle parted into 180 portions, and every of those divisions in 3, or rather 6, smaller parts. . . . The index of that instrument, with the sights, etc., are not unlike to that which the square hath : In his backe prepare a vice or scrue to be fastened in the top of some staffe, if it be a circle, as heere : let your instrument be so large that from the center to the degrees may be a foote in length, more if ye list, so that you not erre in your practices."

For steep upward sighting, he used an artificial horizon.

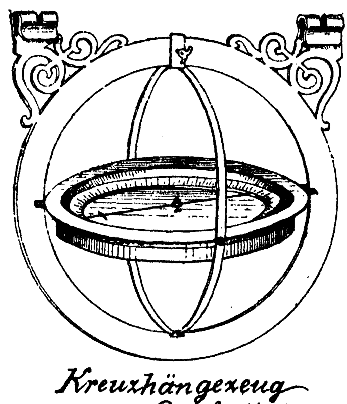
In the same year Diggs published his *Stratiaticus*, in which he says that while he had access to certain of Roger Bacon's

* This derivation is given by Stanley in his work on *Surveying Instruments*. Bauernfeind says (vol. i., p. 288): "It cannot be said with certainty how the word 'theodolite,' as applied to angle-measuring instruments, originated. It was used in England as early as the sixteenth century, and probably had its origin there. Prevailing opinion, formerly, assigned its derivation to two or three Greek words, one of which was λίθος (stone), and basing it upon this derivation the word should be written *theodolith*. But more recent archæological research proves that any attempt to associate the word as we now have it with the Greek is a mistake, as it is more probably a corruption of 'the alidade' by the English from *al'idade*, the Arabic term applied to the radius of the astrolabium."

unpublished MSS. he discovered a letter from his father, describing a method of "viewing distant objects by placing perspective-glasses at due intervals." This was certainly an application of the principles of the telescope, which he, no doubt, like others, had discovered by personal research and experiment.

The period of the casual invention of the telescope is involved in some obscurity. Though there is ample evidence that the ancients of Ovid's time knew something of it, its introduction as a philosophic instrument probably belongs to Friar Bacon, who conducted his experiments in Paris, and died at Oxford in 1294. Its construction and uses were handed down through the generations as a secret, like all other "works of iniquity" that aimed at an advance in science. Later, in 1590, when Jensen, the spectacle-maker, showed his improved instrument to Prince Maurice, he was required, under severe penalty, to divulge no information concerning it, so that only the prince should be aided by it in his warfares; but Galileo, having had it described to him in 1608, constructed at Padua a telescope of three diameters' power, and presented it to the Doge of Venice. It was not until about this year that the opticians of Holland made the practical application of the telescope possible, and inaugurated a new era in the science of astronomy.

FIG. 6.

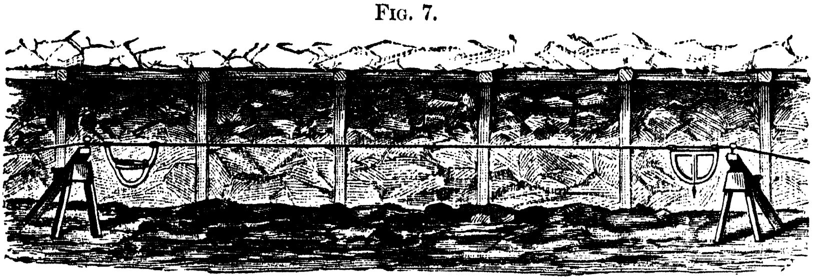


The first systematic and exclusive treatise on mining engineering was the *Geometria Subterranea* of Nicholas Voigtel (Eisleben, Saxony, 1686), in which the methods and instruments described exhibit, after a lapse of 130 years, a natural development yet small improvement over those of Agricola. In fact, mine-surveys were conducted on the continent, and probably also in England, by Agricola's primitive means, until Balthazar Rössler, in 1633, invented the method of suspending from a taut hempen cord a gimbal-compass (Fig. 6) and clinometer, by which the magnetic bearing, incli-

nation and length of any course were at once determined with comparative ease.* The accuracy of this system, which Voigtel describes, depended largely upon the perfection of graduation, the precision possible in reading the clinometer, the catenary curve of the cord on long courses (augmented by the weight of the instruments hung upon it), and the surrounding attractive influences upon the magnetic needle; but it is certainly the first method for the determination of the angular value of precipitous grades without correction for mechanical imperfections in the apparatus.

In 1681 Thomas Houghton published a small treatise upon subterraneous surveying in the Derbyshire mines,† in which he described a use of strings, plumbs and compass very similar to the method of Rössler, except that the *dial*, in a long rec-

FIG. 7.



Surveying by Rössler's Method.

tangular box, was applied by hand to the side of a string, held by two persons, and afterwards measured with a rule.

The method of Rössler, or some adaptation of it, prevailed throughout Europe with remarkable tenacity up to the beginning of this century; and even to-day, at some mines, no other instruments are used; while at others the use of these, in conjunction with the theodolite, is not infrequent. For about eighty years the prestige of the method was undisputed, though it underwent various modifications to suit the conditions of practice. Hempen cords gave way to brass chains; but these (like Gunter's, of 1620, which was substituted for Houghton's rule in English collieries) were found to elongate by tension and friction, so that frequent adjustment became necessary. The catenary curve of the cord, chain or wire was always a matter of

* *Die Entwickelung der Markscheidekunst*, M. Schmidt, Freiberg, 1889.

† *Rara Avis in Terris*, T. Houghton, London, 1681.

perplexity until about 30 years ago, when Prof. A. von Miller-Hauenfels, of Vienna, deduced rules for the suspension of the clinometer in positions to indicate the exact grade between the two stations.

In 1775 Hofrath Kästner designed a quadrant-clinometer, which was suspended from the ends of an index-arm bearing a vernier-scale. The plummet was still used, but only for the purpose of insuring the verticality of the zero-point.*

In 1877 Schneider designed a complete circle of aluminum, dispensing with the plummet entirely, and substituting an alidade with opposite verniers, the verticality of which was determined by a bubble on its lower arm.†

The hanging-compass also underwent various reforms in fruitless attempts to employ it successfully in the presence of iron. Up to 1749 it had not been materially changed from its original construction. The works of both A. Beyer, of Altenberg, and F. W. von Oppel, of Dresden, published in that year, contained nothing new in instrumental construction; but each introduced the use of sines and cosines in the calculations.

In the second edition of Beyer's work, as revised by Lempe in 1785, appeared, for the first time, an illustration of the now common form of hanging-compass (see Fig. 7), said to have been made by Schubert of Freiberg.

The most notable modifications of this instrument are comparatively recent, and include the adjustable forms of Braunsdorpf (1834), Lendig (1846), Reichelt (1856), Osterland (1860), Lehman (1873), Plamineck (1878), Fuhrman (1879), and Penkert (1880).

In the earliest times mine-plans were rare and rude. The object of most surveys was to retrace on the surface the contour of a subterranean opening. "Underground," says Houghton, "the dial is guided by the string, but on the surface the string is guided by the dial." But as the importance of maps became more obvious, the hanging-compass was so modified that the compass-box might be removed and transferred to a brass protractor-plate, where it was clamped in exact position, and the survey was plotted with the same instrument used in making

* *Lehrbuch der Praktischen Markscheidekunst*, O. Brathuhn, p. 34.

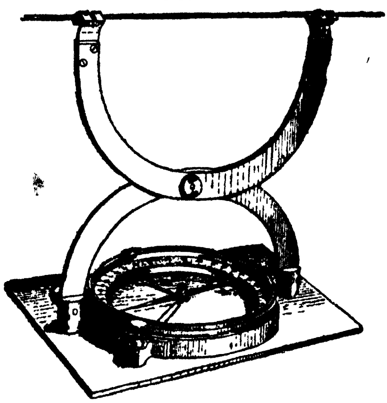
† *Oesterr. Zeitsch. für Berg- und Hüttenwesen*, 1877, p. 367.

it. This method is described in the first edition of Voigtel's work (Eisleben, 1686), and is also spoken of in *The Exact Surveyor* by J. Eyre (London, 1654) as though it had been customary many years previously. Describing the circumferenter of that day, Eyre says:

"For portability this instrument exceedeth any other, and is usually made of wood, containing in length about eight Inches, and in breadth about four Inches, and in thicknesse three quarters of an Inch, the left side whereof is divided into divers equall parts, most fitly of twelve in an Inch, to be used as a scale of a protractor, the Instrument of itself being fitting to protract the plat on paper by help of the Needle, and the degrees of Angles, and length of Lines taken in the Field."

The idea is creditable; but its benefits are questionable, in view of the fact that the magnetic influences are not the same in the office as in the mine. Moreover, for plotting-purposes, the delicacy of a 3-inch needle in a circle graduated to only $\frac{1}{2}°$ is not beyond reproach. Since 1801, the protractor-plate itself has been so provided with adjustable tangent-semicircles (see Fig. 8) that it could be used for both purposes; but, though widely used by mining-captains in Germany, it does not permit very accurate work.

FIG. 8.



The Compass-Protractor as used for Mine-Surveys.

The constancy of the magnetic needle has been questioned only in times which must be considered as recent, compared with the long period of its use; but long before angular or trigonometrical surveying had been presented as the only rational method, the variable susceptibility of the needle to magnetic influences had been the subject of investigation and discussion important to the mining engineer, who had no alternative in localities of strong attraction but the use of the very instrument most affected thereby. The earliest astronomical observations to determine the secular variation were made in Paris in 1541, when the needle pointed 7° E. of N. By 1580 it had reached a maximum of 11° 30′, when it began to recede, reaching the

true meridian in 1666. The yearly variation then became westward until 1814, when a maximum of 22° 34′ was attained!

Bennett H. Brough says:

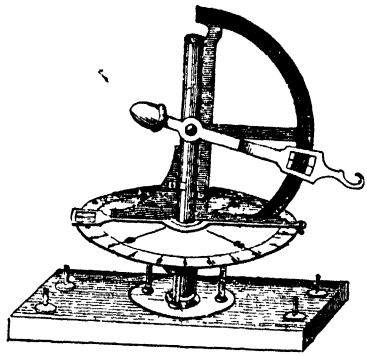
> "There can be no doubt that in times past a neglect of this variation has led to errors involving great loss and serious danger. . . . . Regular observations were not made until the middle of the seventeenth century; but there is a passage (which, however, is so obscure that its meaning is doubtful), apparently referring to the declination of the needle, in the oldest treatise on mining, an extremely scarce work, written in German and published in 1505. No copy of the first edition of this 'well arranged and useful little book,' as the anonymous author calls it, is known to exist."

In 1763 Isaac Prince, of Bonsal, in his *Miner's Guide or Complete Miner*, says, with respect to magnetic surveying:

> "The knowledge of ye quantity of this Declination, which is pretty near ye same one year as another and sometimes differs very little for many years together, enables us to adjust ye Needle in such a manner as if it had no Inclination at all. Though ye knowledge of this Inclination has hitherto been fruitless, it is to be hoped yt some time or another some advantage or profit may be discovered by its regularity."*

Annual and diurnal variations were also studied and dealt with as intelligently as the time and place permitted; but general efforts to remedy the erratic and deceptive demeanor of the needle in the presence of iron finally resulted in the invention and introduction in Germany of the *Eisenscheibe* or iron disk. The first forms of this instrument were described in the third edition of Voigtel (1713), though L. C. Strum, of Frankfort, had proposed the use of the astrolabium for the miner as early as 1710.† Fig. 9 represents the design of J. G. Studer, of Freiberg, which is somewhat of an improvement over the original forms, though its principal features are the same. The disk, as will be noticed, was graduated, like the compass of that day, into

FIG. 9.



Eisenscheibe. [illegible]

* Quoted from *Cantor Lectures on Mine-Surveying*, B. H. Brough, London, 1892, p. 9, in *Surveying by the True Meridian*, E. W. Newton, F.G.S., Falmouth, 1895.

† *Vier Kurz Abhandlungen*, Leon. Chr. Strum, Frankfurt, 1710.

twenty-four hours, the twelfth hour marking the N. and S. cardinal points. In conducting a survey by its use, two, and preferably three, instruments were employed; one being set up at each adjacent station. The indicator-arms were then tied together with a stout cord, first on one side, then on the other, observing the interior angles. In the same way the angles of inclination on the vertical arc were noted; and then the last instrument was brought forward to establish a new station. Later, each hour was subdivided into fifteen equal parts, so that each division corresponded to a degree of the sexagesimal system, making it, as compared with the compass, a most reliable instrument for this work; indeed, it is con-

FIG. 10.

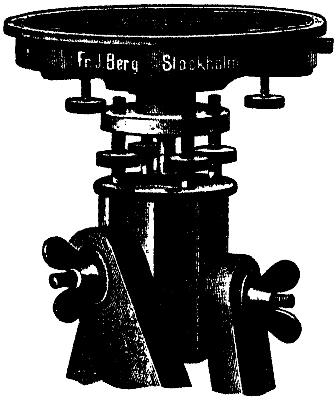
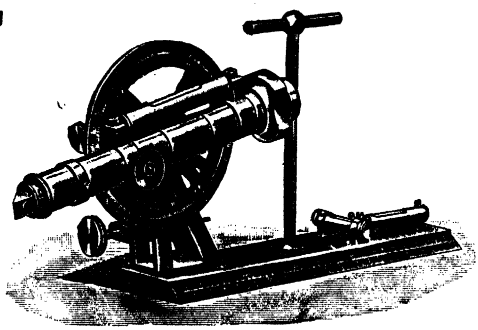


FIG. 11.



Modern Swedish Mine-Tripod and Alidade.

sidered by German authorities to be the predecessor of the perfect mine-theodolite now in general use.

The same feeling seems to have prevailed also in Sweden, where we find mining engineers, near the beginning of the nineteenth century,* discarding the compass entirely in their magnetic iron-mines, and substituting the graphic method of conducting mine-surveys by plane-tables of a peculiar make, which, with rare exceptions, has been in use ever since.

The invention of the plane-table is generally attributed to Prætorius in 1537; but Leonhard Zubler, in the first published account of it (1625), credits its origin to Eberhard, a stone-mason.

* *Handladning uti Svenska Markscheidereit*, Horneman, Stockholm, 1802; also, *Reise durch Skandinavien*, J. F. L. Hausmann, Göttingen, 1811–19, vol. v., pp. 115–126.

Who introduced it into Sweden for mine-surveys, Prof. Nordenström is unable to determine; but the earliest instruments, no doubt, were of rude construction, with only a sighted alidade. I present here (Figs. 10 and 11) a modern tripod, showing the clamping-ring by which the paper is held in position while receiving the plot. The telescopic alidade does not differ from those in general use in other countries for surface-work, except that its vertical circle is full, reading to minutes, and the base-rule that carries the bubbles has the linear subdivisions engraved along the edge.

The Rapid Traverser of Henderson, of Truro, Cornwall, introduced in 1892, is very similar to this mode of construction, except that its alidade is pivoted at the center, instead of being free to move in any position. The survey cannot be plotted in the field, as by the Swedish method. Disks of celluloid, or preferably white enameled zinc, are employed, and the direction of each course or draught is marked upon one of the five concentric circles engraved upon its surface. This instrument and method, it seems, will eventually supersede in Cornwall such magnetic surveys as caused the recent casualty in Wheal Owles mine at St. Just.

The magnetometer of Prof. Robert Thalén of Upsala and that of Tiberg are the only other Swedish instruments that have come to the writer's notice. Thalén's is called a simplified modification of the magnetic theodolite of Dr. Lamont;* but in reality the association is very remote, consisting only in the so-called sinus-method of using it, which has been borrowed from Lamont.† It is a simple compass-instrument, having a magnet upon an arm in the line of sight, so arranged that, by its application and removal at the proper time, the azimuth of each course is obtained.

Tiberg's (1880) is almost identical with this, except that the compass-box is set in bearings upon low standards, and occasionally made to revolve in a vertical position for use as a dip-needle, in determining the location of magnetic ore-deposits.

In *Wärmländer Annaler* of 1888, Mr. Sjogren describes some

* *L'Industrie Minière de la Suède*, G. Nordenström, p. 22. (A translation of the methods here described occurs in *Mines and Minerals*, Scranton, Pa., November, 1898.)

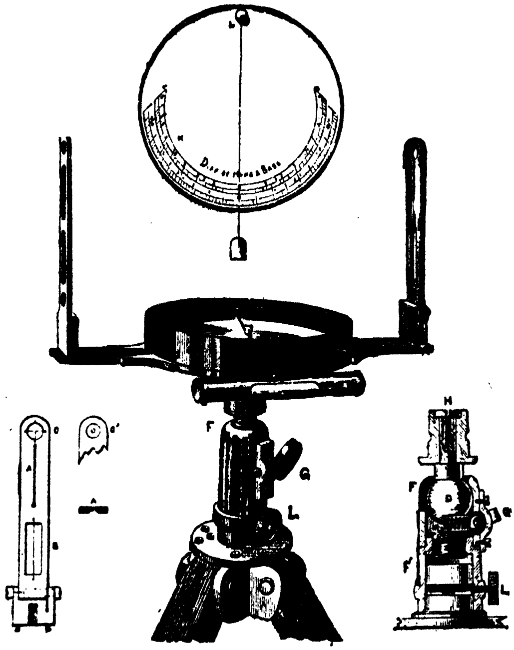
† *Sur la recherche des Mines de fer à l'aide de mesures magnétiques*, R. Thalén, 1877.

very creditable work of this kind, done with Tiberg's instrument at Persberg.

In England the magnetic needle and Houghton's methods were adhered to, amid recurring failures and disasters, with a loyalty that prevails even to-day in some parts of Cornwall.*

In 1778 Dr. W. Pryce declared that methods similar to Houghton's were still in vogue.† He says:

FIG. 12.



Old English Miners' Dial.

"The instruments used are a compass without gnomon or style but a center-pin projecting from the center of the compass to loop a line to, or stick a candle upon, fixed in a box exactly true and level with its surface, about 6, 8 or 9 inches square, nicely glazed with a strong white glass, and a cover suitable to it hung square and level with the upper part of the instrument; a 24-inch gauge or two-foot rule and a string or small cord with a plummet at the end of it; a little stool to place the dial horizontally; and pegs and pins of wood, a piece of chalk, and pen, ink and paper."

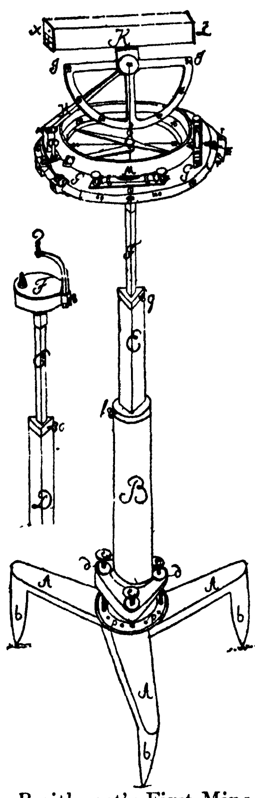
Later (about 1785), extended sights were added; the little three-legged stool no doubt became the tripod; and, with one or two other slight improvements, we have in England, just

* *Proc. Royal Cornwall Polyt. Soc.*, 1893.

† *Mineralogia Cornubiensis*, W. Pryce, London, 1778, pp. 202-213.

before the beginning of this century, the type of dial shown in Fig. 12, in a modernized form, as made by W. F. Stanley, of London. Who is responsible for its conception as a whole is not known—probably no one particular person, as too many years were required to bring it to even its original simple construction; but Adams must have improved upon it; since an instrument similar to this is described in his *Geometrical Essays* of 1803. It originally had no bubbles, but was considered level when the needle floated freely.

FIG. 13.



Breithaupt's First Mine-Theodolite.

The early makers were, however, in the habit of partly counter-sinking two spirit-levels in the compass-box if desired; but these were not recommended, because, by reason of their small size, they were seldom accurate, and the operator could not test them, or even adjust them if they were known to be untrue.

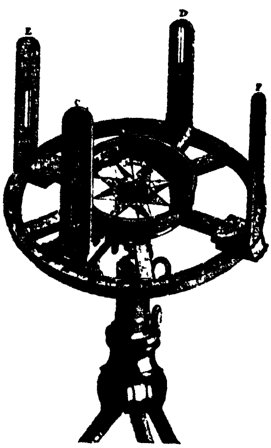
In the English dial shown in Fig. 12 the socket is slotted down on one side (F) so as to permit the limb to be turned vertically, making the sights horizontal. In this position, after the long bubble beneath the compass is made level, the compass-box cover is adjusted and the very small plummet, suspended from its top by a hair or silk thread, is made to read zero on the graduations. The sights can then be tipped up or down, and gradients up to 45° can be determined approximately. It will be noticed that the graduated cover, as in most other English dials, has the correction for declivity marked upon it, so as to save the operator any calculation in this particular. Lean began this practice in Cornwall in 1825, receiving a prize of thirty guineas for his borrowed improvement.

In 1798 H. C. W. Breithaupt, of Cassel, introduced an instrument which may be justly designated the first of mine-theodolites. Fig. 13 is reproduced from the original drawings of the inventor by the courtesy of his heirs, F. W. Breithaupt & Son.

In his book,* Herr Breithaupt described the use of this instrument in a new system of surveying, which he himself first practised in the Reichelsdorfer copper-mines of Hesse.

Jesse Ramsden had already (1760) constructed a circular dividing-engine in London, in response to the urgent demands of geodetic engineers for circles of greater accuracy than those heretofore graduated by hand; and M. Pierre Vernier, of Burgundy, had long since (1631) published in Brussels a description of the micrometer-scale which now bears his name. We must note here, parenthetically, that Vernier's scale was adjusted by hand until Helvetius, the celebrated astronomer of Danzig, invented, about 1650, the clamp-and-tangent movement. Applying these precedents, Breithaupt circumscribed a compass with a carefully divided circle, read by Vernier-plates; invented an arrester that should clamp the needle when not in use; superimposed an adaptation of the clinometer that was surmounted by a sighting-tube; and supported this compact combination upon a sort of telescopic tripod-stand, which could be adjusted for height by means of set-screws at the side.

FIG. 14.



The Jones Circumferenter.

Instead of using two or more instruments, as was customary in employing the *Eisenscheibe*, he designed a signal-lamp, two of which were used interchangeably with the instrument. These instruments, which sold for 8 carolin ($33.70), he made himself, and felt the necessity of economy so strongly that he made a plain sighting-tube to take the place of a telescope.

In the same year Prof. Guiliani, of Vienna, constructed a mine-instrument which he called a *Katageolabium*. Like the *Graphometer Souterrain* of Gen. Komarzewski, it was closely allied to the *Eisenscheibe*.

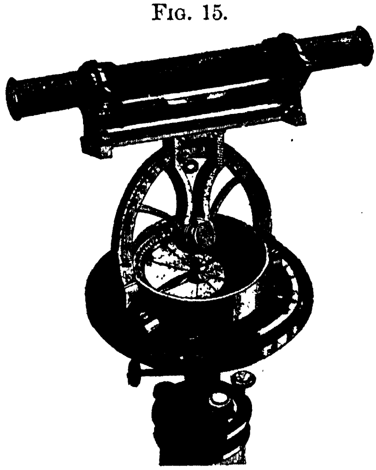
In England the same spirit of economy and consequent simplicity of construction has always prevailed. In 1796

* *Beschreibung eines neuen Markscheidinstruments*, Kassel, 1800.

W. & S. Jones, of London, introduced a circumferenter (Fig. 14), which, in addition to the ordinary compass, was provided with a 10-inch brass circle, divided into single degrees and read by the "*nonius*," as they were then called in England, to 5 minutes of arc. In offering this instrument to the engineering profession, Mr. Jones said: "The error to which an instrument is liable, where the whole dependence is placed on the needle, soon rendered some other invention necessary to measure angles with accuracy; among these the common theodolite, with four plain sights, took the lead, being simple in construction and easy in use."*

It was fitted with one pair of fixed and one pair of movable sights, like Henderson's dial (1869), and "marks the date," says Mr. Newton, "of the first attempt in England to conduct underground surveys, in the presence of iron, with any degree of accuracy."

Fig. 15.



Simple English Theodolite of last Century, now commonly known as Lean's Dial.

Elliott Bros. made such an instrument for Fenwick in 1822,† but on account of its expense this construction was not much used until Lean began to employ vernier-circles in 1836 in the Cornish mines.

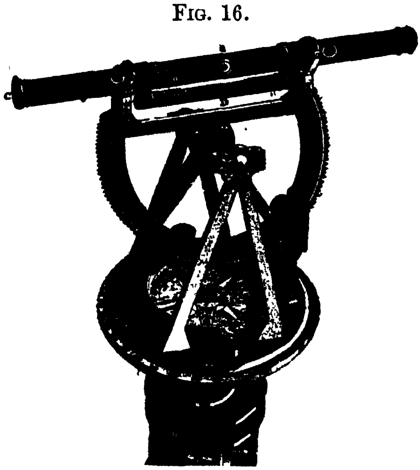
So far as available evidence can be relied upon, what is now commonly known as "Lean's dial" was the first telescopic mine-instrument ever introduced; but there is some doubt concerning this fact. Fig. 15 is taken from the *Geometrical and Graphical Essays* of George Adams, published in 1797. It is there said to be intended as a fair sort of cheap theodolite. Lean's grandson can furnish no authentic information; but it is known that when it was first used in mine-surveys, and made for Captain Lean by the Wilton Works of St. Day, the vertical arc and telescope were removed for common sights and replaced only for surface-work. In accordance with

* Adams's *Geometrical Essays*, revised by Jones, London, 1797, p. 223.

† *Treatise on Mathematical Instruments*, J. F. Heather, London, 1849.

the prevailing custom, the sights alone were used underground. The dial was set up only at alternate stations; the back- and fore-sights were read with the needle; and the bearings were assumed to be correct. The centers of vertical shafts simply became intermediate stations; but if an inclined shaft was encountered, a cross-staff was set up in it, so that one pair of sights were directed at a candle-light in the bottom; then the dial was set up in a drift and made to bear upon the other pair. In this way the magnetic bearing of the shaft was calculated by adding or subtracting 90°. "Many miles of dialing have been done," says Franklands, "by this rapid but blindfold method, with no means of closing the survey." It was not thought possible, at that time, to connect the surface and underground surveys by anything but magnetic bearings, so that the telescopic attachment to the so-called Lean dial can hardly be considered the antecedent of Breithaupt's first telescopic mine-transit (made in 1832), though it has been widely used in more recent years for such purposes. Since 1871, E. T. Newton & Son, of Camborne, have made the Y's of the telescope interchangeable with the arc or the sights. By thus mounting the telescope upon the limb just over the compass-box, the instrument becomes a substitute for the Gravatt level, which has found great favor in the English colonies.

FIG. 16.



Seven-inch English Theodolite of Last Century.

It is a recorded fact that the general design of this dial came from the standard model English theodolite, to which it bears a strong resemblance. This instrument is rarely used in English collieries on account of its cost; but its construction has long since been such that vertical sights from one side were possible. The prolongation of an inclined shaft alignment, however, can be accomplished only by reversing the telescope in its bearings or by revolving 180° upon its horizontal limb.

Why it is that English engineers adhere to this model has always been a source of wonder to the American profession; for, in the relationship of the telescope to the vertical circle, it is one of the most extreme of eccentric types. Its retention, however, must be well merited, or it would have been supplanted by the transit-principle of Ramsden in 1803, or by the concentric model of Sir George Everest (1837), which practically became obsolete twenty years ago. But whatever it has to commend or to disqualify it, the important developments in English engineering-instruments must be followed with this theodolite. In Gardener's *Practical Surveyor* (1737) we have descriptions of this theodolite, much improved and brought to nearly its present form by Jonathan Sissons, an optician of London (Fig. 16).* It was not perfected, however, until 1760, when Ramsden sensitized its graduated "brain," and John Dolland sent pure light coursing through its telescopic "soul." Dolland discovered the construction of an achromatic telescope in a compound objective of two kinds of glass (in direct antagonism to the principles laid down by Sir Isaac Newton), by which both spherical aberration and errors arising from varying refrangibility were in a great measure overcome. While this discovery was very important in the manufacture of geodetic and astronomical instruments, "the most perfect objectives were not made," says Thomas Dick, "until after the improvements of Dr. Blair, of Edinborough, Rodgers, of London, and Fraunhofer, of Munich, in the first quarter of this century"†—just before telescopic instruments came into general use for mine-work.

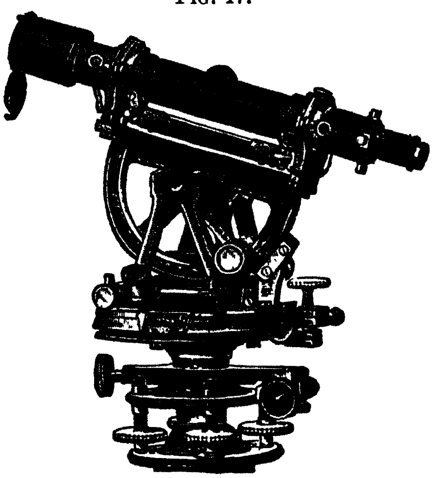
Lean's dial, then, might have had a practically achromatic telescope. It might also have been provided with a diaphragm and cross-hairs; for Huygens discovered that any object placed in the mutual focus of the two lenses of a Kepler telescope (1611) appeared as distinct and well defined as any distant body. Following this established theory, in 1669 Jean Picard, Marquis Malvasia and others crossed silken fibers in the mutual focus of their astronomical instruments; and these, while generally acknowledged to be too large for the work required, were

* This particular figure represents an instrument made by Ramsden, Jones, Adams, and others, after the general style of Sissons.

† *The Practical Astronomer*, Thomas Dick, LL.D., New York, 1846.

used, in lack of something better, for a century or more. In 1755 Prof. Fontana, of Florence, proposed the use of spider-webs, though it is said they were not put into practical use until Troughton secured for this purpose the webs of the geometrical spider, at the instance of David Rittenhouse, of Philadelphia, who was then constructing the first American telescope.

FIG. 17.

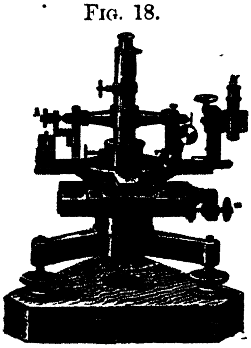


Modern English Theodolite.

At the time the telescope begins to play some part in the construction of mining instruments we find it, so far as possibilities are concerned, a very perfect device; but as the use of fine-quality lenses entailed considerable extra expense, and as almost any contrivance was considered good enough for surveys in dirty little underground passages, we find at that time generally only poor-quality and low-power telescopes in use.

For the precise shaft- and tunnel-work involved in the construction of the Great Western Railway in 1843, Bourne was probably first to use the high-class English theodolite, shown in Fig. 17, in connecting the underground- and surface-surveys through the vertical shafts.

FIG. 18.



Hassler's Instrument with Perforated Vertical Axis.

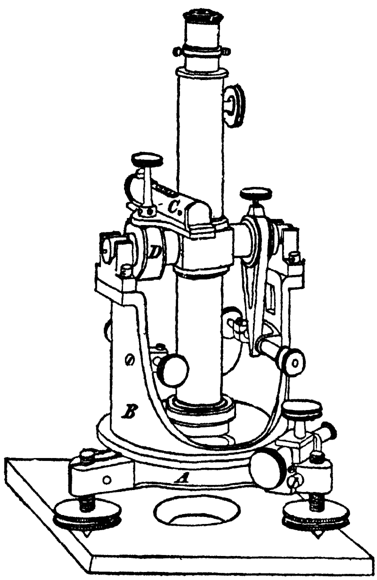
Since that time, and doubtless before it, the ideal instrument for nadir-sighting has been considered one in which the vertical axis of the concentric type should be enlarged and perforated sufficiently to permit such observations.

In 1824 F. R. Hassler, Superintendent of the U. S. Coast Survey, designed such an instrument (Fig. 18), which, as subsequently improved by General Ibañez, was pronounced by the European Degree-Measuring Commission to

be perfect for the purposes intended.* It had two slow, independent lateral motions, like the sliding stage of a microscope, by which the cross-hairs could be brought exactly over a point in the aligned base-line to insure perfect parallelism in the subsequent setting of the metallic measuring-rod. As its use was restricted to geodetic engineering, it has no special application here, but is inserted only to establish the priority of the invention.

Prof. Viertel first used the telescope of an eccentric theodolite for conducting surveys in vertical shafts, by suspending a plummet through the diopter; but as the instrument was not steady enough, and the plummet was centered only with great difficulty, the method was abandoned as impracticable.

Fig. 19.



Nagel's Nadir-Instrument.

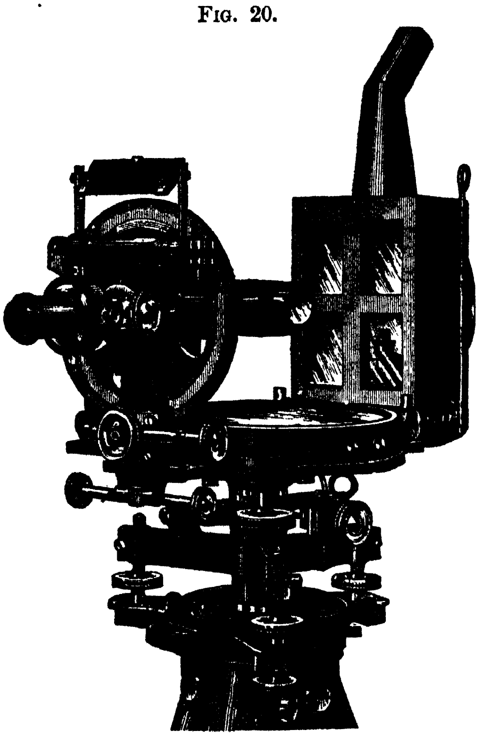
Prof. A. Nagel, of Dresden, had a nadir-instrument (Fig. 19) constructed without vertical axis, which could be centered over a shaft with great precision by means of a center-plug in the base, which was afterwards removed to leave the opening free for the purpose designed. The adjustment of his instrument was the same as that of any ordinary theodolite. To obtain a true vertical sight he first set a plate of mercury under the telescope somewhat below its focal distance. Its surface must necessarily be a perfect horizon, and under a fair illumination will reflect the image of the cross-hairs. When they are brought exactly to coincide with this reflection, the optical axis of the telescope will be truly vertical if it is originally in perfect adjustment. It was his practice to revolve the telescope 180°, and repeat the operation, rectifying any defect at the bottom of the shaft, upon a mechanical stage which was pro-

* *Die Landmessung*, Dr. C. Bohm, p. 605.

vided with co-ordinate micrometer-scales to determine the mean projection. The experiments of both Viertel and Nagel were recorded in *Der Civilingenieur* of 1878.

In 1876 H. D. Hoskold submitted plans to Troughton & Simms for a similar instrument; but the plans having been lost in Paris, its introduction in England was allowed to lapse.* This firm, however, in 1885, devised an instrument (Fig. 20) that is very remarkable among the instruments of its class. Its *single* axis of revolution was perforated by a hole ⅝-inch in diameter, which provided a sufficiently large opening without the necessity of increasing the size of the base, so as to be heavy or out of proportion.

FIG. 20.



Troughton & Simms Prismatic Nadir Dial.

The 4½-inch needle was supported at the top of the perforation upon a sort of little flat trident, and was never removed for nadir sighting. It absorbed, of course, some rays of light, but did not interfere with an otherwise perfect view. The broken telescope, which was focused by movement of the ocular, was so placed that its prism came directly over the perforation, and vertical sights could be made by setting the vertical limb to the zero of its vernier, which, in turn, was insured for verticality by the long bubble upon its arm. This vernier was illuminated by the reflection of a mirror, so placed above it as to deflect the light from the large lamp opposite, which was designed to counterbalance

* *Trans. Am. Soc. Civ. E.*, xxx., 153, 1893.

the weight of the telescope and vertical circle. The outer circumference of the compass-ring was graduated, and could be read to minutes by opposite vernier-plates that were rigidly attached to the compass-box. The circle was first placed at zero and held by the pin, with looped string, shown protruding below, and the telescope moved in azimuth by means of the large thumb-screw, also shown below the compass-box.

The instrument had no plummet. In ordinary traversing the instrument was set up at any convenient position, and the station-point sighted in afterwards through the axis.

Its makers volunteer the information that there never has been much of a demand for this instrument, and that it is now no longer made. This was, perhaps, because of the inconvenience of the broken telescope in general work and the conditions that prevented observations in dips greater than 45° and less than 90°.

All such instruments have generally been allied with the straight-line or tunnel-transits. In 1877 Buff & Berger made such an instrument for G. H. Crafts. The base, having the shape of a horseshoe, was mounted upon three leveling-screws. This instrument was used for the shaft- and tunnel-work of the Dorchester Bay sewer, for the city of Boston, and afterwards to re-establish the boundary between Massachusetts and New York.*

In 1887, E. A. Geiseler, now Assistant United States Engineer at Savannah, Ga., designed plans for a nadir-instrument in which both the graduated circles and compass-box were to be retained. He proposed that the vertical axis be slightly increased in diameter and perforated with a hole $\frac{3}{32}$-inch in diameter. Its upper orifice was in the base of the compass-box and ordinarily contained a plug, so designed as to support the needle. When vertical sights were necessary, the plug with needle was carefully removed and set just to one side, by means of a lever-arm operated from outside the compass-box. The invention received editorial comment in the *Engineering News* and *Railroad Gazette* at the time, but was not executed until the following year, when the F. E. Brandis Sons, of Brooklyn, constructed an instrument on these prin-

* *Jour. Ass. Eng. Societies*, vol. iii., No. 9, 1884.

ciples, with some modifications to suit the ideas of their customer. The perforation was made 1¼ inches in diameter and the compass-box was dispensed with entirely. In its place was supplied a trough-box compass suspended from the horizontal axis in a manner similar to that described in the works of Prof. Weisbach.* The plummet was suspended from a hinged plate below, which, when released, permitted a clear vertical sight with the main telescope. The length of the vertical axis was reduced, owing to its large diameter; but this did not make a well-proportioned lower base, and the instrument never was duplicated.

Fenwick, of Durham, was the first to introduce in England (in 1804) a general system of mine-surveying by observing only the magnetic bearing of the first course. He points out in his work that as mines are now becoming more skillfully and economically developed, a more scientific system of engineering is essential. "The general use of the magnetic needle," he says, "in subterraneous surveys has been found to be a great source of error on account of ferruginous substances, which exist in all mines, attracting the needle; whence, in general, old surveys and plans are found to be extremely defective." For these reasons he suggests that, except in beginning the survey, the use of the needle be abandoned, and declares it much to be regretted that the general use of the compass still prevailed and mapping was conducted on the original diagrams through scores of years without any consideration for secular variation. He further says: "If the student be acquainted with the application of spherical trigonometry to astronomy, he will find the following method of finding the true meridian to be greatly preferable . . . ." But herein lies the secret of the survival of the compass, which could always be relied upon as indicating a relative direction. What did most of such men as were engaged in digging ore from the earth know or care about the sciences, or how to consult and interpret the Ephemeris?

It was lack of education and a want of the means for the diffusion of knowledge, as exemplified in our colleges and institutes of to-day, that made these primitive, simple and unreliable methods so enduring. Fenwick no doubt found his adherents;

* *Die Neue Markscheidekunst*, J. Weisbach, Braunschweig, 1859.

but in the early part of this century dialing with the needle alone predominated in England, as well as in America, where such engineering as was necessary was performed with some foreign type of dial or hanging-compass.

In recent years the hanging-compass has been re-designed by Queen & Co., of Philadelphia, and is said to be still indispensable to certain surveyors in Virginia and Pennsylvania.* The excuse for employing the hanging-compass in cramped and tortuous channels to-day, however, seems absurd; for the transit can be made to do the most reliable work, even when removed from the tripod, anywhere a man can take it.

Fig. 21.

Draper's Mine-Transit.

The earliest American instruments were descendants, as it were, like the American people, of English ancestry, with an infusion of German and French influences, which have ever since combined with natural American skill to make them the resultant of the most approved appliances and methods.

William J. Young, established in Philadelphia in 1820, introduced the first American type of transit in 1831, probably after that of Ramsden (1803), who introduced the transit principle in small English theodolites at that time. The earliest American engineers objected to the intricacies of the ordinary English theodolite for reasons already stated; but "most of these," says Mr. Young, "who had only local training, could not understand the superiority of vernier-circles over the compass-sights for seeing past or around a tree or other obstruction!"

Young's first transit was graduated to read by vernier inside the compass-box to 3′; the needle was 5 inches long, and the telescope 9 inches long and of low power.

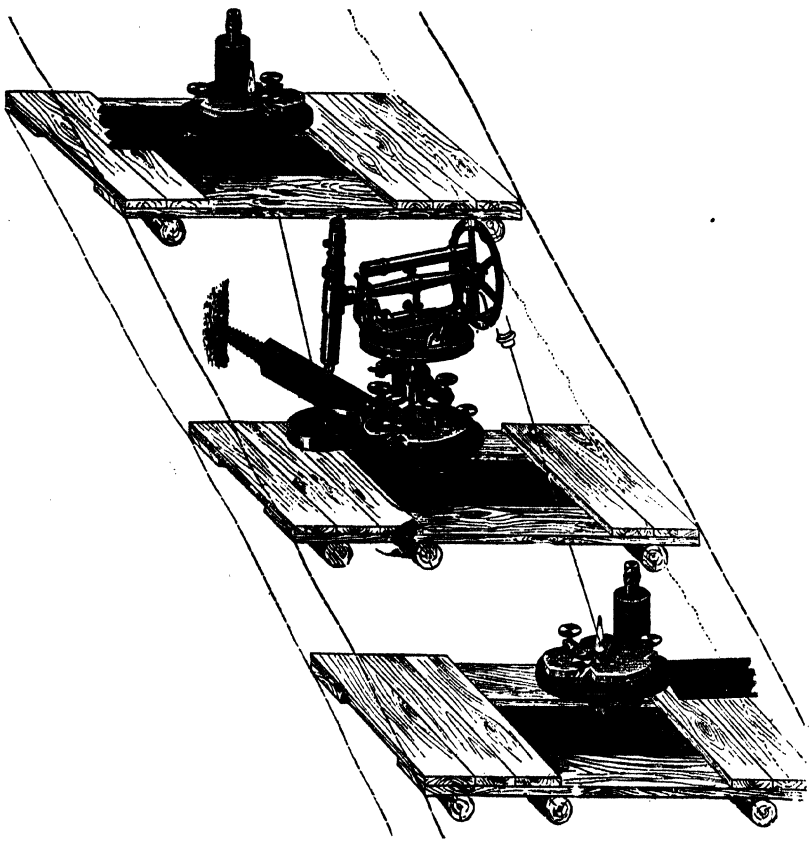
The first distinctive mine-transit that ever appeared in Amer-

* *Eng. and Min. Jour.*, vol. lii., p. 125, Aug. 1, 1891.

ica was introduced by Edmund Draper (established in Philadelphia, 1815), about the year 1850. Fig. 21 is reproduced from a photograph kindly loaned by F. C. Knight, his successor, who says:

"It had full vertical and horizontal circles, each of which was read to minutes by one vernier. The upper horizontal plate was extended somewhat beyond the

Fig. 22.



Borchers' Eccentric Instrument, in an Inclined Shaft. Instrument and Lamps Supported on "Freiberg Brackets."

compass-box, ending in two piers that supported the pillars of the telescope. The open space between them permitted true vertical sights or the observation of any angle between the horizon and nadir, without any correction for eccentricity beyond the mere mechanical addition of the telescope's distance from the instrument's center to the base component. The telescope's power was 16 diameters."

Considering the times, the design of this pioneer instrument

is both original and praiseworthy, and in some respects is superior to the eccentric model of Prof. E. Borchers (Fig. 22), which he introduced in 1835 for special surveys in inclined shafts. Zenith instruments in Germany had for some time previously been constructed with eccentric telescopes, but had never before this date been used in mines. In his work on mine-surveying* Borchers says that while the ordinary theodolite (since 1832) has been employed for most mine-work, this instrument is most convenient for inclined shafts, though not intended for magnetic observations.

The first instrument of this kind was made for him by Breithaupt. Both circles were 16 cm. in diameter and graduated to read 20″. The hub of the telescope was rigidly attached to a flattened extension of the horizontal axis by four screws, and the optical axis was made to be exactly at right angles to the axis of revolution by lateral movement of the small cylinder that contained the eye-piece and diaphragm. The intersection of the cross-hairs was then made to coincide with the optical axis thus established, independently, by the usual methods.

He used an artificial horizon in taking steep upward sights to an 84° limit, and recommends it as being convenient and portable, but his enthusiasm never became contagious. Observations were made first on one side, then on the other; a mean was deduced, and the true inclination was calculated, without any correction for eccentricity, which in the longest courses amounts to only two or three seconds. One of the most interesting features about this instrument is the mountings, known since the improvement of Prof. Dr. Schmidt, in 1882, as the "Freiberg brackets," upon which the instrument and signal-lamps rest without being clamped. They have taken the place of the tripod to a considerable extent in Germany, for, without much effort, they can be screwed into the timbering and perform duty in low places where tripods could not be set up. The stand had been made of wood until 1885, when wood was exchanged for metal that would not wear.

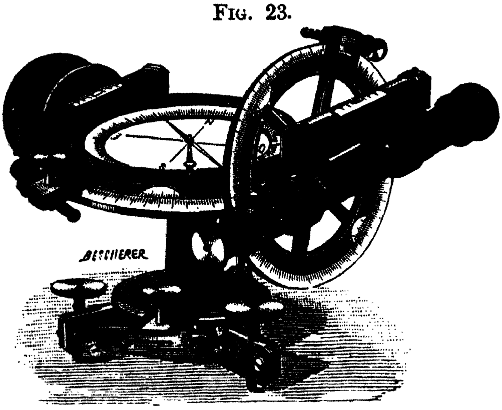
In that year Prof. Chrismàr, of the Schemnitz School of Mines, introduced a support consisting of two hollow wrought-iron pipes, with steel teeth at the outer ends, and sliding one within

* *Die Praktische Markscheidekunst*, E. Borchers, Hanover, 1870, p. 131.

the other, in such a way that they could be firmly clamped between drift-timbers at distances varying from three to seven feet, and not interfere with the passage of tram-cars or trucks. The theodolite-stand was then clamped to the center, making a combined weight of sixteen pounds.

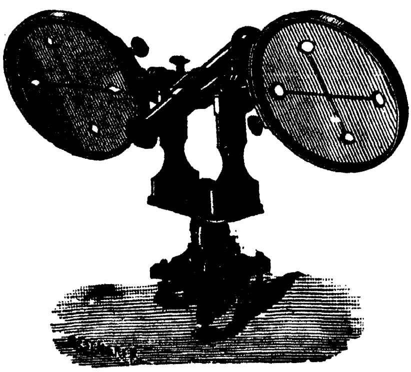
This was similar to the support of Kawerau (1862), whose staves slid vertically side by side, being clamped by girders. To this the bracket was attached, and arranged to move by jointed arms in three directions to facilitate centering.

FIG. 23.



Combes's Theodolite, with Eccentric Signal Targets.

FIG. 23A.



In 1845 Prof. C. Combes, of the Paris School of Mines, introduced in France a modification of the eccentric theodolite which has been in general use there ever since, until in late years it is being replaced by the concentric telescope; for however well this or any other such instrument may be constructed, the number of parts and the eccentricity of the telescope, which must be counterbalanced by a weight, are permanent causes of derangement. Fig. 23 shows a 16 cm. circle, modernized pattern, built by H. Morin, of Paris; but "the original," says M. André Pelletan, "had no compass, but a large horizontal circle graduated to read 30″, surmounted by a delicate striding level, as in Borchers' theodolite." This style of mounting the telescope prostrate upon the vertical circle was first practised by Prof. Morin, of Paris, in 1634, when he attached a telescope to the movable index of a graduated arc for the purpose of measuring the diameter of fixed stars.

It is customary in Germany, France and other foreign countries to use with the eccentric theodolite a twin target, which operates interchangeably with the instrument. It is called *Doppelsignal* by the Germans and *viseur de mine* by the French, and is so constructed that each target is as far removed from the center of the signal as the telescope is from the center of the instrument. This provides against any necessity for correction in reading horizontal angles, whether the telescope is used on one side or the other, normally or reversed. In setting it up, upon its own tripod, the assistant, after leveling it by means of the box-bubble between the standards, directs the sighting-tube at a light held at the instrument.

Combes's instrument, as well as all others in France, at the option of the engineer, is graduated by the sexagesimal system common in America and most European countries, or by the centesimal system, in which each quadrant contains 100 grades (as adopted in 1801 at the suggestion of Laplace when the metric system of weights and measures was proclaimed legal and compulsory).

The division of the circle into 400 degrees for compasses and mine theodolites is still of doubtful benefit, as the minute spaces are made to correspond to only 32.4 sexagesimal seconds, while the centesimal seconds are so diminutive as to be practically impossible in small circles.

A want of uniformity in denominate nomenclature also tends to retard its progress. The logarithmic tables of Borda (1801), Plauzolis (1809), Gauss (1873), and Gravelius (1891), are all at variance in this respect; but the proposal of Prof. S. Jordan in 1891 to write, for instance, 24^{g} 86^{c} 50^{cc}, seems to deserve general adoption.

In America the decimal system of graduation was introduced by S. W. Mifflin, C.E., for construction-work on the Pennsylvania Railroad, after the methods of M. Minot, engineer for the Orléans Railroad, who in 1856 popularized in France that system of railroad engineering known as tacheometry.

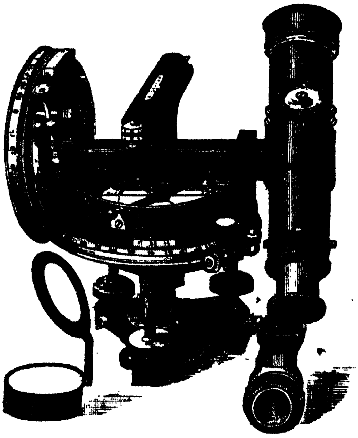
For about thirty years, Young has made transits in which one vernier reads the sexagesimal circle in sixtieths, etc., and the opposite one in tenths and hundreds; but except, perhaps, for railroad work, in which the tangential deflection of curves can be laid out with facility, the French system of graduation

has never met in America the favor accorded to the metric system of weights and measures.

While considering the eccentric type of mine-theodolite, it may be well to notice here, though not in exact chronological order, its introduction and application in England.

In 1869 Louis Casella, of London, introduced a small portable instrument for Alpine and military surveying (Fig. 24), by which good results have been obtained, not only in the determination of astronomical time and latitude, but in shaft-work in the British mines.

FIG. 24.



Casella's Portable Theodolite.

Its circles are only 3-inch, graduated to read minutes, the vertical circle being placed opposite the telescope, to aid somewhat in making the equipoise perfect. It is mounted upon the tribrach locking-plate, introduced by Everest with his theodolite in 1837, and since possessed of greater popularity than his instrument. This miniature theodolite, weighing only 3½ pounds, with the pocket mine-theodolite of Breithaupt (1869) with 8 cm. circle, having a total weight of 1.7 kg., and the aluminum mine-transit of Keuffel & Esser, weighing only 2.1 kg., are about the smallest instruments we have to deal with. The only strong argument in their favor is their portability, but efficiency ought not to be sacrificed in this way for a novelty.

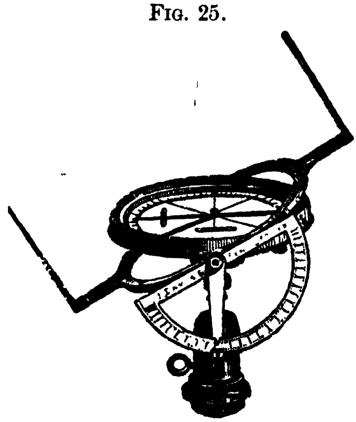
J. Winspear, of Hull, in 1870, made an eccentric instrument for Hoskold, to which was given the distinctive name of *Angleometer.* It differed from Casella's model in that the compass-box was raised somewhat above the horizontal plates so that the axis, connecting the telescope on one side with the vertical circle on the other, should pass between. The long axial bubble being then placed on the side nearest the vertical circle, the compass was left perfectly unobstructed, as we find it in Prof. Combes's model. Combes's theodolite was the lowest form, because, in placing both telescope and vertical circle on one side, it had no need of a horizontal axis.

The *Angleometer* received an award at the London Exhibition of 1872, where there was also displayed with it a plain compass-dial, with plain sights on one side and semicircle opposite.

The use of the eccentric telescope has by no means become general in mine-surveying in England, but important and very satisfactory work has been conducted with it. Horizontal angles are usually repeated, first on one side, then on the other, to obtain a mean reading; but if only on one side, in platting, a small circle is drawn to scale about the station equal in radius to the eccentricity of the telescope, and the courses laid out tangentially to this by use of the English system of platting with the parallel ruler.

Within the present century greater improvements in mine-surveying methods and the instruments used in conducting them have been achieved than in all the previous history of the world; but by far the greater share of this century's progress has occurred during the past fifty years. A hundred years ago an engineer could err occasionally with good and sufficient reason. To-day there is absolutely no excuse for anything but perfect results.

FIG. 25.



Hedley's Dial.

America and Germany have been perhaps more largely instrumental in perfecting the apparatus used in mine-surveying in this period than England, though the latter is entitled to a just share of credit in an endeavor to perfect the construction of the dial or circumferenter, so as to obviate the necessity of using the more expensive theodolite.

In 1850 John Hedley, H. M. Inspector of Mines, being convinced of the inconvenience of the vertical arc on Lean's dial in obstructing frequently a clear reading of the needle, caused John Davis, of Derby, to construct an improved model (Fig. 25). Its principal feature was the swinging limb, by which vertical sights up to about 50° could be observed and read upon the index of the vertical arc at the side, leaving the face of the

compass quite unobstructed. With the Hedley dial, the miners' chain of 10 fathoms or 120 links still continued to be the means of linear measurement, and gave the early surveyor nearly as much trouble in attempting to correct for its cumulative and compensating errors as for the vagaries of his needle. It was (and is still) made with the ten end-links of brass,* and when used for measuring is laid on the floor and each chain-length marked with a piece of chalk, the entire course being represented by so many fathoms, feet and inches.

In chaining up slopes the plumb-line was still used, as on the surface. This is the very earliest method of conducting measurements on inclined planes, and gave rise to the practice, yet widely followed in Europe, of conducting surveys in inclined shafts by successive plumbing and leveling. The several lengths of the plumb-line are recorded as the vertical components, and the several distances from the base of one plumb to the point of suspension of the next make up the horizontal aggregate.

In France and Germany special appliances have been made to perform this work. That of Mr. O. Cséti, of Hungary, is no doubt the most recent.† Briefly, it is a small leveling telescope fastened by a sliding and clamping collar to a hollow square rod of 0.67-inch section and 5 feet long, that may be suspended from any station. From the top downward the rod is graduated to cm., while the vernier on the sliding collar determines vertical distances with great precision. Lengthening-bars are also supplied. Its total weight is 5 kg.

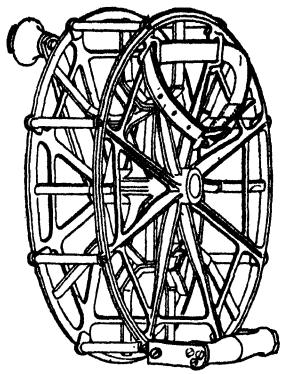
While speaking of the English chain, it must be noted here that the chain of Rittenhouse, which comprised 80 links or 66 feet, was quite generally used in American mines until Eckley B. Coxe and others started a reformation, some twenty-five years ago, in favor of the steel band that has now practically consigned the chain to President Cleveland's "innocuous desuetude." In 1874 Dr. R. W. Raymond said: "While so much improvement in recent years has been made in mine-instruments, the chain remains unaltered. Nothing can be inherently more objectionable, as a standard of measurement, than a chain composed of links that wear by friction,"‡ and, we may add, connecting-rings that elongate by tension.

* *Colliery Management*, J. Hyslop, Wishaw, 1870, p. 23.

† *Berg- und Hüttenm. Zeitung*, vol. liv., p. 391, Nov. 8, 1895. ‡ *Trans.*, ii., 224.

In the first American steel tapes the points of graduation were marked by impressed figures on white solder, or by small brass rivets; but etched graduations and figures now predominate. The 500-foot tape used by the writer is one made by Keuffel & Esser, of New York, and exhibited by them at the World's Fair. It is $\frac{2}{10}$ of an inch wide, graduated at every foot throughout its entire length, and wound on a gun-metal reel, the convenience of which will appear obvious by inspection of the illustration, Fig. 26.

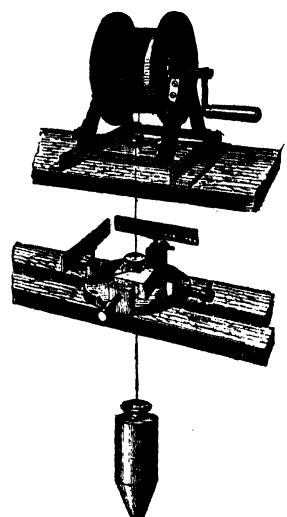
FIG. 26.



Skeleton Reel for Long Mine-Tapes.

With the Hedley dial went the continuance of the practice of connecting the surface- and underground-surveys by magnetic observations, which operation was greatly facilitated by its rocking limb. In 1856, however, Arthur Beanlands, C.E., undertook a purely astronomical method of accomplishing this result by use of the theodolite alone; but the observation of stars from the bottom of a vertical shaft was attended with such difficulty that he projected an alignment, first from above, then from below, by the use of lights, with excellent results.*

FIG. 27.



Schmidt's Centering Apparatus.

Thirteen years before, Thomas Baker, C.E., had suggested and used a method (held in derision by colliery-surveyors at that time) of suspending from a straight-edge two fine copper wires by heavy weights in mercury, and completing the survey with the theodolite and interchangeable tripods and targets.†

The accuracy of this method has always been somewhat impaired by the extremely short base from which to work, and the inevitable

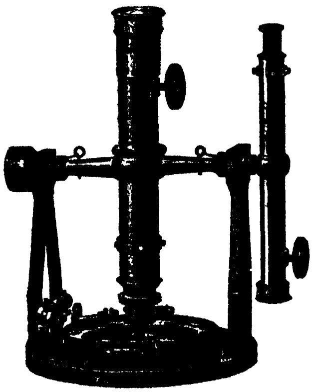
* *Trans. North Eng. Inst. M. E.*, vol. iv., p. 267.

† *Subterraneous Surveying*, Fenwick and Baker, 1888, p. 40.

oscillation in the wire; but the centering-apparatus of Prof. Dr. Schmidt (1884), Fig. 27, has regulated the method to a nicety. The vibrations are carefully measured by repeated observation on co-ordinate scales, and the wires are finally clamped in exact mean position. The instrument is then ranged into their alignment, and presents a means only less accurate than the later method of suspending one wire in each of two vertical shafts, thus securing a length of base limited only by the distance between the shafts.*

In England, Beanlands is usually credited with having first used the theodolite to connect the underground and surface-surveys; but Bourne's method of 1843 and Borchers' of 1835 certainly are entitled to precedence. In fact, we may nearly always look to Germany for the beginning of all important steps in the advancement of mining engineering.

FIG. 28.



Early American Method of Mounting Side-Auxiliary.

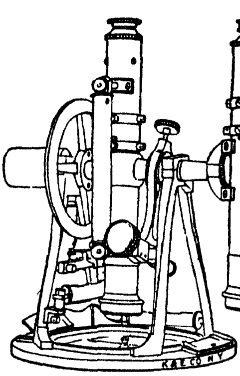
The eccentric telescope of Borchers was found so convenient for the purposes intended that it was employed by American engineers as a side-auxiliary, which could be attached to concentric instruments at will, and removed when not in use. The first American types were of simple construction. The horizontal axis of the main telescope was perforated with a hole large enough to permit a spindle, conjugate with the hub of the auxiliary, to be inserted and held fast by pins (see Fig. 28), such as are used in the Y's of a level.

Adams, in his *Geometrical and Graphical Essays* of 1791, says: "In the present state of science it may be laid down as a maxim that every instrument should be so contrived that the observer may examine and rectify the principal parts; for however careful the maker may be, it is not possible that any instrument should long remain accurately fixed as it came from the manufacturer." But we find in these first forms (see, for instance, Fig. 29) no means of testing the adjustment of the side-auxiliary; and the startling fact must be recorded that,

* *Methoden der Unterirdischen Orientirung*, M. Schmidt, Berlin, 1892.

almost up to the present time, the adjustment of all auxiliary telescopes has had to be assumed as correct upon the guarantee of the maker, at the risk of their working loose on their bearings.

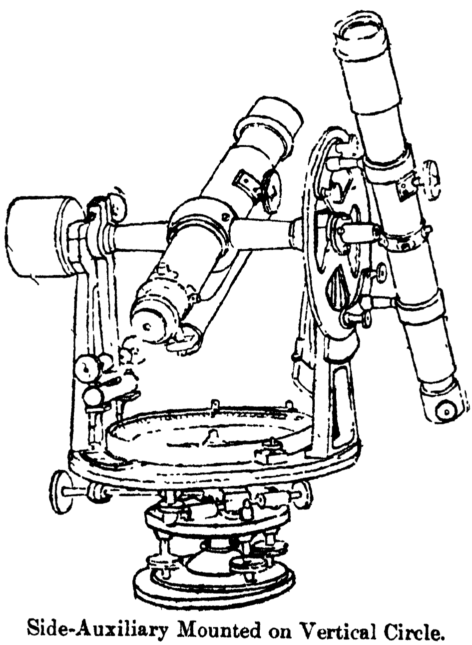
FIG. 29.



Keuffel & Esser's Concentric Instrument, with Side-Auxiliary.

When side-auxiliary telescopes first came into use in America, new instruments were always equipped in the manner just described; but when engineers who had been using simple concentric instruments caught the contagion, they sent their transits to the factory to receive this valuable adjunct. In such cases it was customary with the house of Young to mount the side-telescope upon the vertical circle by means of set-screws and clamping-plates, in the manner shown in Fig. 30. In the event that the instrument had not been provided

FIG. 30.



Side-Auxiliary Mounted on Vertical Circle.

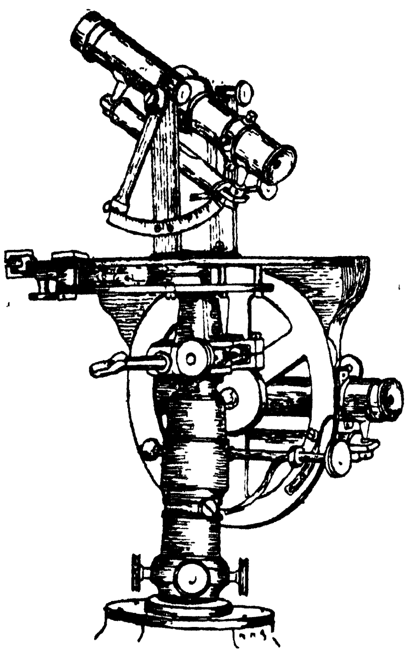
originally with a full vertical circle, one was permanently attached to an improvised prolongation of the horizontal axis.

The adjustment of such an appliance was no easier that when it was mounted on a spindle, but the method presented the advantage of holding the auxiliary more firmly in the position in which it was placed.

The shifting tripod-head, shown in Fig. 30, was made by Young in 1858, and has been ever since a most valuable convenience to mining engineers.

As the German method of mounting eccentric telescopes had its influence upon the early manufacture of American side-auxiliaries, so also we may notice the effect of the French types in the model (now obsolete) presented in Fig. 31, which was known as the "Lake Superior" pattern, and used in 1858, when the copper-mines of that region first became valuable.

FIG. 31.



Lake Superior Pattern.

The upper plates were not unlike those of an ordinary transit.

The smaller upper telescope was intended only for general work, and provided simply with a short 60°-arc and loose vernier-arm, as first made by Young in 1850.

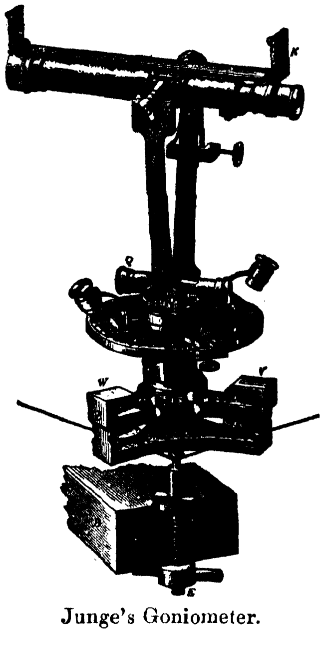
This style of vernier may be clamped in any position, and by repetition made to read any angle up to 90°. The vertical plate was similar in shape to the horizontal, to which it was attached. It was always at hand for vertical observations, but its additional weight, its exposed position and delicate construction, conspired to insure the rejection of this instrument in favor of the regular concentric types.

In the meantime, some progress had been made in Germany with a view to supplement, at least, the hanging-compass with such instruments as could be made to verify it, and even to

supplant it. In 1861 Prof. Junge, of Freiberg, invented his *Goniometer** (Fig. 32).

The inventor apologetically announces: "This instrument will be found a great convenience for use with compasses in mines, if, however, it shall not furnish some means of attraction;" but being himself convinced of its superiority, he recounts among its advantages: First. All that can be accomplished by use of a compass can be determined by the goniometer. Second. It can be used anywhere in the mine, and even to great advantage in inclined shafts, if the conditions are not too cramped. Third. The accuracy in reading angles is more pronounced, and gross errors cannot be committed, etc.

FIG. 32.



Junge's Goniometer.

The horizontal circle was small, as compared with the height of the standards, and graduated to read minutes of arc. The instrument had no vertical circle, as the old method of cords and suspended clinometer was adhered to, with the early mode of setting up instruments of this class. A plank was wedged between the walls, and a hole bored in which to set and clamp the spindle of the instrument, as illustrated more fully in Fig. 33. This system was first used and taught by Prof. Lang von Hanstadt, of the Schemnitz Bergakademie in Austria, as early as 1835,† and was subsequently recommended by Prof. Weisbach in 1859. In this modern system here portrayed, the instrument and targets are of equal height, and are made interchangeable in the 3-legged base by opening the set-screw at the side. In this way the target may be removed, the conical spindle of the instrument set into the socket, and the tangent-screw of the azimuth-axis fitted to work upon the pin shown protruding from the right leg. This system of set-

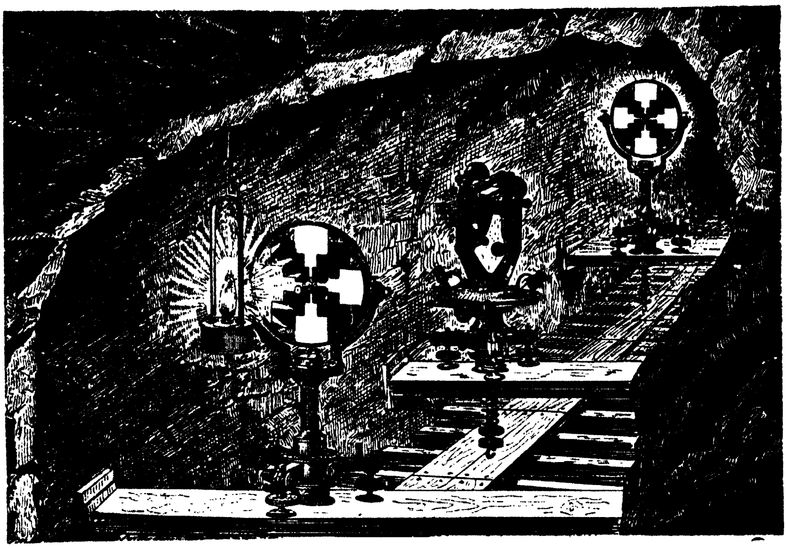
* *Die Geometrischen Instrumente*, C. G. K. Hunäus, Hannover, 1864.

† *Anleitung zur Markscheidekunst*, J. N. L. von Hanstadt, Pesth, 1835.

ting-up is not so complicated, delicate or expensive as the Freiberg brackets, but takes a little more time to secure, and cannot be used in large openings, any more than the other can be used in rock-tunnels where there is no timbering.

The telescope of Junge's goniometer revolved completely in its standards, and could be made to sight objects downward in declivities unusual with most theodolites. If steep upward sights were necessary, the eye-piece prism was used and attached to the ocular by means of a spring-clamp. The tele-

FIG. 33.



Breithaupt's Modern Mine-Theodolite with interchangeable Targets and the *Spreitzen* system of setting-up.

scope was provided with two diopters for ranging it into line, though for universal observation these preceded the telescope and must be looked upon as the forerunners of extended sights.

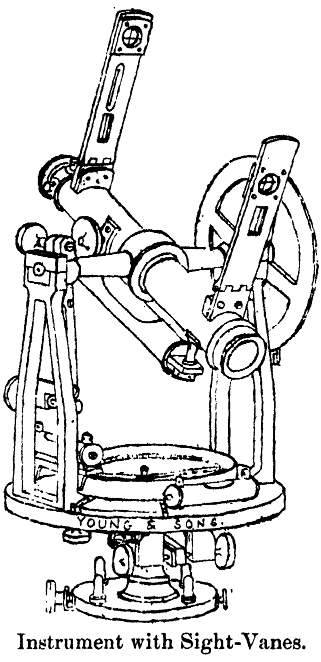
The origin of sight-vanes must be sought in the very earliest times, and, as applied to mine-surveying, in the first instruments ever used. When they had outgrown their usefulness as principal factors, they were used in conjunction with the telescope, first to assist in directing it upon an object, and later, being lengthened and provided with windows, to make observations in dips that approached verticality. When and by whom they were first used for this purpose the writer is now unable to

say; but the instrument shown in Fig. 34 was built by Young at an early date and shipped to Mexico. The tripod upon which it was mounted was one of America's first adjustable-leg forms, and was very heavy and awkward.

Diopters and extended sights had been used for observing very near objects that came within the focus of old-time telescopes; but their later application for vertical sighting gave rise in America, no doubt, to the top-auxiliary telescope.

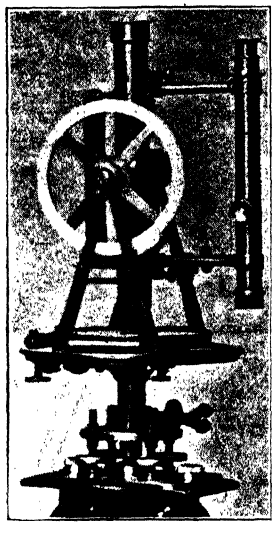
The available evidence concerning the invention and intro-

FIG. 34.



Instrument with Sight-Vanes.

FIG. 35.



Draper's Top-Auxiliary.

duction of the top-auxiliary telescope is not conclusive, but Knight says that Draper was probably the first to introduce it in 1840. The instrument shown in Fig. 35 is easily identified as Draper's by the peculiar style of tripod-head. The base-plate of the instrument is set upon two threaded spindles projecting from the tripod-head, and held in position by two milled nuts.

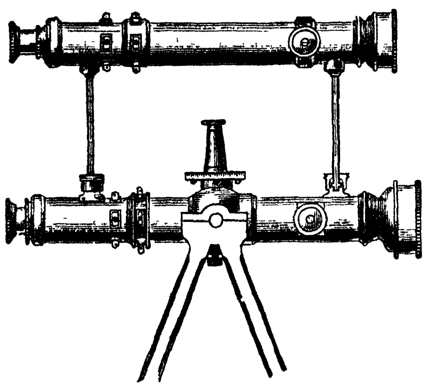
Gurley's first top-telescopes (Fig. 36) were attached to the main by coupling-nuts and "steady-pins" which provided for their ready removal and replacement. Until very recent years

these have been made by all American makers in styles similar to the first model, with no means of effecting adjustment for parallelism and alignment beyond the guarantee of the maker, which, in the best makes, could not be relied upon as permanent.

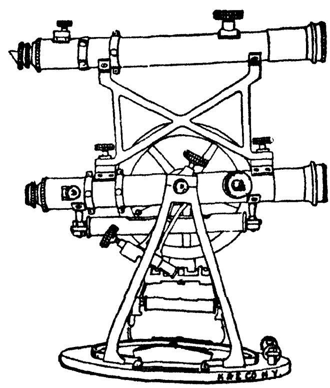
So far as the writer knows, no very serious errors of execution have been recorded during the fifty years' use of the non-adjustable auxiliary, though there has always been room for such error in the fact that it was hardly possible to place the instrument a second time in the exact position it had once occupied.

The growing sentiment among American engineers in favor of instrumental construction permitting accurate adjustment led

FIG. 36.



Gurley's Top-Telescope.



A German Improvement.

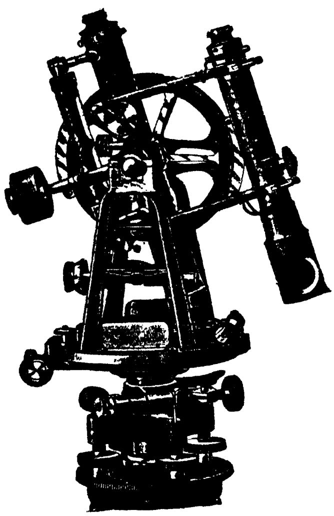
Per Larsson, E.M., then at Vulcan, Mich., to begin in 1882 a reconstruction of the top-telescope by providing at least that it should be capable of most of the adjustments of a level, by placing it in Y's that were permanently fixed to the main telescope. Such an instrument was made for him by Buff & Berger, of Boston, and was then justly considered very complete, inasmuch as the line of collimation of the top-telescope could be adjusted independently by rotation in its Y's. The adjustment for parallelism, however, was too complex an operation to be undertaken outside the factory.

In the later models the Y's were put on top, so as to be partly balanced by the long bubble beneath; but in Mr. Larsson's instrument they were put under the telescope, so as to be

out of harm's way, and worked very satisfactorily. The later model shown in Fig. 37 is provided with the gradientor-screw, as applied by Prof. Stampfer, of Vienna, in 1873, to the tangent movement of the horizontal axis of transit-instruments. As employed in the rapid determination of distances it was first used by him in 1839, and mentioned in his work of that year.*

The silvered head, appearing in Figs. 37 and 38, is graduated into 50 parts, and the screw is cut with such a value as to cause the horizontal web of the telescope to move, for each graduation of the head, over a space of .01 foot upon a rod 100 feet from the focal center of the telescope. In mine-instruments this arrangement is very convenient, light and efficient for the establishment of water-courses, contours, etc.

FIG. 37.



Larsson's Top-Telescope.

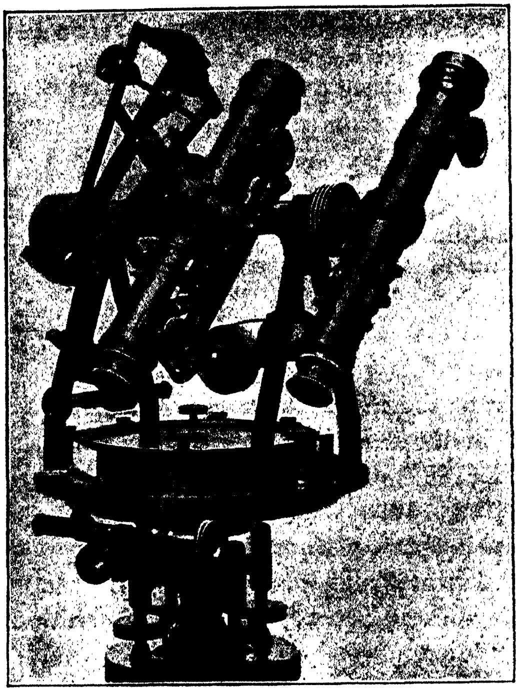
In Europe special instruments are constructed for this purpose; the *Distanzmesser* of Stampfer, the gradiometer of Stanley, and Short's telemeter-level (1889), being notable examples. Since 1894 Casella has applied the principles of Short's telemeter to the miners' dial; but the instrument is without provision for the observation of anything but gentle gradients.

Somewhat earlier, the manner of attaching the side-telescope was also improved. The transverse axis of the main telescope was extended to end in a threaded hub, upon which the side-auxiliary was screwed with a coupling-nut, very similar to the Gurley method of securing the top-telescope, and, we may say, attended with very nearly the same difficulties. After attaching it loosely, and revolving the auxiliary by hand until its horizontal-wire should cut the same point with the main telescope, it was a matter of some perplexity to tighten the hub while it remained in this position, and a matter of speculation

* *Elemente der Vermessungskunde*, C. M. Bauernfeind, München, (1856 to) 1890, pp. 417, 422.

whether it would long remain that way. Fig. 38 shows an admirable feature in what are known as "disappearing stadia." These were invented in 1880 by Hon. Verplanck Colvin, Superintendent of the New York State Survey, to overcome the liability of error in leveling with the wrong hair. He caused Gurley to put the stadia-hairs upon a separate diaphragm, so as to be entirely out of focus when not in use. They are very

FIG. 38.



A Modern Gurley Mine-Transit with Solar Attachment and "Disappearing Stadia."

desirable in all instruments, and particularly so in mine-transits, where so much depends upon correct vertical angles. As early as 1865, B. S. Lyman used glass stadia-rods in the coal-mines of Pennsylvania to avoid chaining through mud.* They might have been used, however, in Europe at a much earlier date by engineers who chose to employ such aids.

In 1778 William Green, a London optician, published an

* *Jour. Frankl. Inst.*, 3d series, vol. lv., 1868, p. 384.

account of the possibilities of "subtense measurement" with either fixed or adjustable micrometer-lines in the focus of the eye-piece. Possibly he got his ideas indirectly from William Gascoign, of Yorkshire, who is said to have devised in 1639 a micrometer for astronomical instruments consisting of two pointers traveling by screw-threads in opposite directions.

No practical use, however, was made of these discoveries until 1824, when Major M. Porro, of the Piedmontese Army Engineers, designed the topographometric instrument with anallactic telescope, which he named the "Cleps." Its first notable application was in the topographical survey of Switzerland in 1836. It is now largely used for military reconnaissances in Germany, being supplied by A. Salmoiraghi, of Milan.

Prof. Baker says: "Stadia-hairs were not introduced in America until after the Civil War, and have not yet come into the general use their merits warrant."*

One other distinctive feature of the Gurley instrument here considered is the solar attachment. This is essentially a Burt solar, but was remodeled by William Schmoltz, of San Francisco, in 1867, and has been mounted since 1874 upon the hub of the main telescope, with trivet-adjustment to secure the verticality of the polar axis. The polar axis-spindle, with hour-circle, is permanently fixed, as shown in Fig. 38, to the main telescope, to which the solar is attached at will, and upon which it revolves, precluding the use of anything but a semicircular vertical arc, which, for the purpose of laying off latitude, is ample.

The original solar compass was invented in 1836 by William A. Burt, of Michigan, as the result of an effort to overcome the annoying defects of the magnetic needle. It was first made for him by Young, and mounted upon a simple ball-and-socket base without tangent-screws. Concerning it Prof. Baker says: "It was a very ingenious instrument, and while, at the time, it deserved the popularity it attained, it now reflects more credit upon the inventor than the one who uses it, as it possesses possibilities of error." But Capt. Talcott, in a letter to Mr. Burt, testified that, in running the boundary-line between Iowa and Minnesota, he could not detect in the line it

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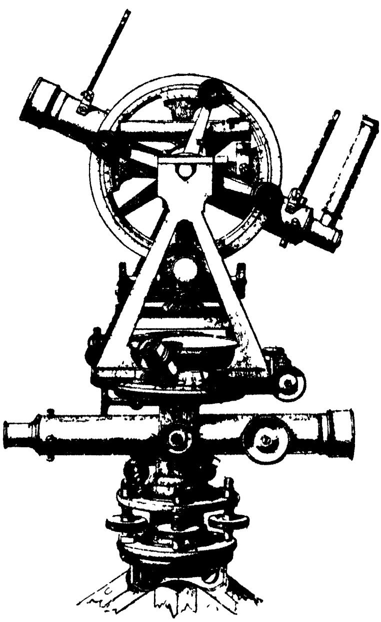
* *Engineers' Surveying Instruments*, Ira O. Baker.

established any variance with the most careful astronomical observations.

Its operation is always more perfect in clear weather; on dark or cloudy days it is impracticable, and when the sun is partly obscured its image is indistinct, leaving room for doubt. In 1878 C. L. Berger proposed a remedy by substituting for the lens-bar a small telescope of low power, but G. N. Saegmuller first made this practical in his excellent invention of 1881, considered on page 729.

FIG. 39.



Hoskold's Miners' Transit-Theodolite.

In the meantime English engineers and mathematical opticians had made such progress as the exigencies demanded and the limitations permitted. In 1857, John Archbutt & Sons, of London, had built for H. D. Hoskold an instrument* (Fig. 39) constructed upon principles prevalent in American types, which are at this day fast superseding the older English models.

In this instrument the upper plates, carrying the standards and compass, were made to project somewhat beyond the horizontal limb, permitting the use of a 4½-inch needle with a 5-inch circle.† The full vertical circle was provided with three vernier arms, and the horizontal graduations were beveled and unprotected; a practice still common in England, but rare in America. The verifying telescope had been in use in the latter half of the last century to insure the stability of astronomical transit instruments, but Hoskold was first to use it with a mine-transit. It had been mounted in small bearings clamped to the

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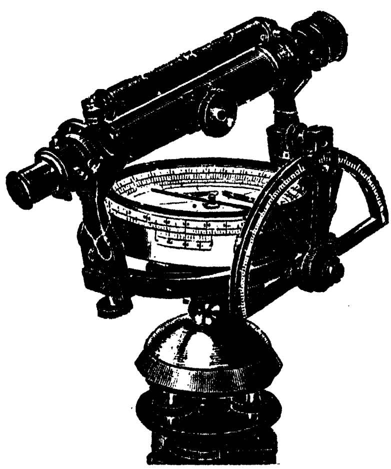
* *Practical Treatise on Mine, Land and Railway Surveying*, H. D. Hoskold, London, 1863; also, *Trans. So. Wales Min. Engrs.*, vol. iv., No. 5, 1865.

† *Prac. Treat. on M., L. and Ry. Surveying*, H. D. Hoskold, London, 1863; also *Trans. So. Wales Mining Engrs.*, vol. iv., No. 5, 1865; also *A Treat. on Mine Sur.*, B. H. Brough, London, 1888.

side of the vertical axis until 1863, when he invented the means of mounting it concentrically, directly under the center of the plates, so that its optical axis should coincide with the zero-line of the horizontal plates. In this way it has been conveniently used for back-sights without stopping at each observation to clamp the zero of the vernier to the zero of the limb. The compound spindles being thus replaced by the verifying telescope, the azimuth axis is inverted between the standards and the vertical axis placed below. An instrument of this kind with two vertical arcs and striding compass was exhibited in the British section at the Columbian Exposition in 1893.

In 1874, two years after the passage of the Mines Act, which regulated, among other things, correct mine-mapping, Stanley, at the suggestion of Mr. W. Preece, mounted a telescope in Y's upon the Hedley dial. The swinging limb still carried the vertical arc, now in an upright position, so as not to interfere with the leveling-screws, and could be inclined to observe angles of depression as great as 55° before the tripod-head interfered. Such angles were determined to the nearest degree only, by a simple index.

FIG. 40.



Telescopic Hedley Dial.

The horizontal circles, however, were graduated to read to 3′, and the 4½-inch needle to ¼°. The eye-piece was inverting, and the telescope had a power of 10 diameters. The instrument shown in Fig. 40, which possibly represents the highest modern refinement in English circumferenters, is provided with the Hoffman quick-leveling head, as modified and improved by Prof. J. H. Harden of the University of Pennsylvania in 1879.* It was first invented and patented in 1878 by Daniel Hoffman, of Pottsville, Pa., and was justly considered an improvement over the designs of Pastorelli (1863) and of Doering (1864).

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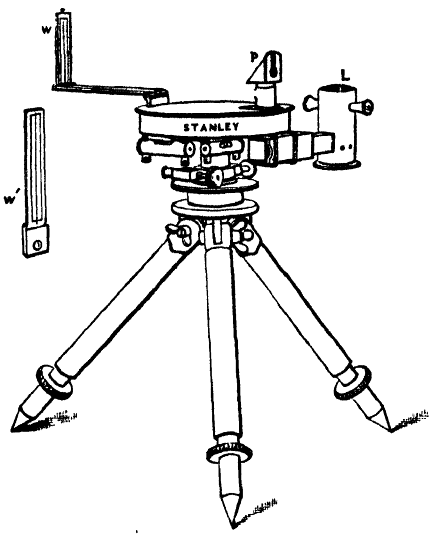
* *Trans.*, vii., 308.

Davis of Derby introduced in 1882 a modification of this improved Hedley dial, in which he made the vertical arc to be replaced by a fixed circular box only 1¾ inches in diameter, with an index-finger traversing the graduated disk. It is necessarily divided very coarsely and made thus compact so as not to interfere with the easy manipulation of the leveling-screws. Its peculiar construction makes it possible to swing the limb in altitude only about 15° each way. Prof. Brough says this is the best instrument for colliery use, and we can only infer that vertical angles must be of little consequence to the English engineer generally. Certain English engineers are also still doubtful about the beneficial use of the telescope in conjunction with the compass for mine-surveying, seeming to agree with Gillespie,* who says "the exactness of the vision of a telescope is rendered nugatory by want of accuracy in the compass and the precision possible in reading the needle," and does not therefore consider it an improvement over the common sights for the ordinary execution of magnetic surveys.

Fig. 41.



Prismatic Compass-Dial.

As late as 1882 Stanley built a prismatic-compass dial (Fig. 41) to conduct surveys in a 30-inch coal-seam. The original prismatic compass was invented by Capt. Henry Cater, about 1814, but this adaptation of it for mine-work is unique. Where space in height is cramped the needle is read from the side. The 5-inch compass has attached to the needle a floating disk of celluloid, upon which the magnetic bearings are read to the nearest ¼° simultaneously with the observation through the sights.

The illumination is effected by prismatic reflection of light

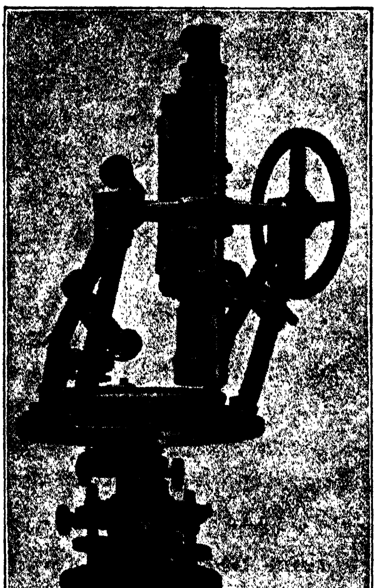
* *Treatise on Land Surveying*, W. M. Gillespie, N. Y., 1856.

from the lamp L, as in the ship's compass. The combined weight of instrument and tripod is 8 pounds. The cylindrical tripod-legs contain screw-threads, which are used for leveling the instrument. This device was patented, but never became popular.

The telescope, no doubt, came into use with the more widespread application of "fast-needle" dialing, as originally employed by Fenwick in England.

The magnetic bearing of the first course alone is observed, and that of all succeeding courses is computed from azimuth survey by use of the vernier-plates. It is then platted by calculated tangents, or by what is considered in England the most reliable of all methods—the use of co-ordinates of latitude and departure deduced from the traverse-tables.* This is a very old practice. In 1791 John Gale published traverse-tables in London, but they had been used from personal manuscript as early as 1635, by Norwood, in the conduct of surveys between York and London.

Fig. 42.



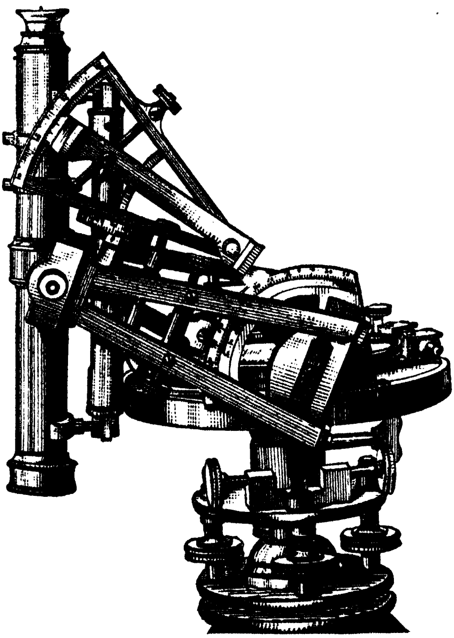
Transit with Inclined Standards.

Returning to the discussion on the Burt solar, it must be observed here that this instrument came into extensive use on Government surveys in the subdivision of public lands. It occasionally took the place of the compass on the ordinary transit; but the revolution of the telescope, and the shadow it cast, interfered somewhat with successful operation. To overcome this difficulty F. R. Seibert, then connected with the U. S. Coast Survey, designed a transit in which the standards were inclined forward, throwing the hub of the telescope just over the edge

* *Mine Surveys*, F. J. Franklands, London, 1882.

of the plates, in a position to leave the solar open without obstruction to the rays of the sun. This instrument was made for him by Young, and was exhibited at the Centennial Exhibition in 1876. For several reasons, however, it was not a success for the purposes intended; but in 1874, at the suggestion of T. S. McNair, of Hazleton, Pa., it was converted into a mining-transit (Fig. 42), which, as such, possessed some interesting features. It was capable of perfect adjustment, and fulfilled in a measure, as did Draper's instrument, the functions of both main and top-telescope. Its construction did not interfere with the direct and perfect reading of horizontal angles,

FIG. 43.



Blattner's "Hinged Standards."

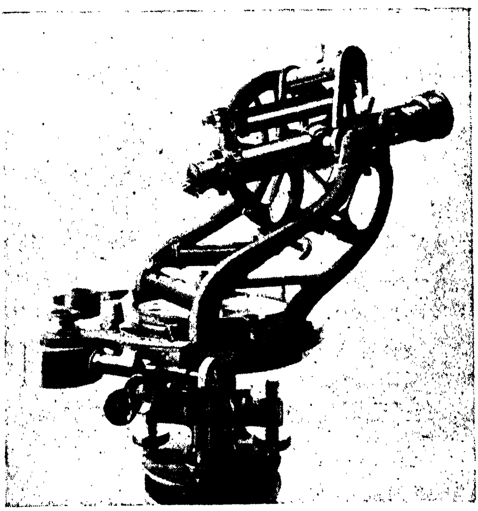
though a counterweight was always required to preserve the equilibrium necessary in the best instrumental construction. The eye-prism is detachable at will. It is inserted between the two lenses of the ocular, forming an image that becomes erect with respect to altitude, but reversed in azimuth. Except by the use of a diagonal eye-piece, a steep angle of elevation can be observed only in a reversed position of the telescope; but its one disadvantage lies in the fact that in the

prolongation of an inclined shaft-alignment it cannot be checked by reversed sights. In such cases one must rely solely upon the perfect adjustment of the instrument.

In 1883, Henry Blattner, of St. Louis, Mo., undertook to overcome this objection by introducing what have since been commonly known as Blattner's "hinged standards" (Fig. 43), though they were not hinged at all. "The vertical arc was of an entirely new pattern," he says, "built very strongly, so as to support the standards in any position of inclination by a clamp and opposing tangent-screws." The solar was mounted upon a pin protruding from the standards in such a manner that the latitude could be laid off on the vertical arc and the telescope permitted to move in altitude independently of the solar. In Blattner's first mineral surveyor's transit, of which accounts were published in the *Engineering News*, the telescope was 7½ inches long, provided with adjustable stadia; the needle was 3½ inches long; and both circles were graduated to read minutes.

Blattner's style of vertical arc, with the method of clamping to it the combined weight of the standards and telescope, was not unlike the English model we noticed in Fig. 40, which, however, presents the disadvantage of having no slow tangent-movement.

FIG. 44.



Batterman's Transit.

All my readers will conclude, no doubt, that any instrument in which the center of gravity of the telescope is so far removed from its support, and depending for its stability upon a clamp, is of defective design; and some will concur with Hoskold in the opinion that, in general, "this mode of construction is clumsy, inconvenient and unsightly."

Such opinions to the contrary notwithstanding, we must

aver that, while no essentially eccentric type is without faults, that of Young (Fig. 42) possesses the least. It may be called the "American Eccentric," and finds favor in many localities.

The most recent design of this type is that built by the A. Lietz Co., of San Francisco (Fig 44). Concerning it, Mr. O. von Geldren says:

"This transit was designed by C. S. Batterman, E.M., of Aspen, Colo., in 1894. The principles involved are not new, but his application of them is unique; and for the work of transmitting a point to extreme positions in the nadir this instrument has given excellent results. It is properly balanced by a counter-weight, and provided also with a socket on a movable arm, in which a candle may be placed in any convenient position.

The eye-piece prism is also attached to a movable arm by which it is kept always at hand without fear of losing it, and can be laid back against the telescope when not in use. The spring key between the leveling-screws assures the immovable position of the base-plates to the tripod-coupling which Lietz has recently introduced. In it the usual thread is replaced by three jaws, constructed upon the principles of the wedge, and locks by friction very firmly into grooves in the base-plates made to receive them. No manner of attaching an instrument to the tripod can be more convenient or safe."

Since 1896, instruments of this class have been constructed upon the "cyclotomic" principle, in which the double compound spindle is replaced by a single axis of revolution, while the admirable qualities of repetition are still preserved by a floating exterior ring, carrying only the numbers, which may be clamped in any position with respect to the horizontal circle. In this way any degree-line may be made the zero-point.

Lietz credits the inception of this idea to Luther Wagoner, C.E., of San Francisco, who found errors as great as 3′ in 90° in instruments the compound spindle-centers of which were not coincident.* The method of repetition, in order to determine the measured arc with greater accuracy, was introduced by Prof. Tobias Mayer, of Göttingen, in 1752, and a little later by Borda, under the name of "Double Repetition or Multiplication." Mr. Wagoner did not wish to abolish this practice, but to introduce a new and safer means of accomplishing the same result.

In 1881, G. N. Saegmuller, of Washington, D. C., who conducts the instrumental establishment of Fauth & Co., introduced

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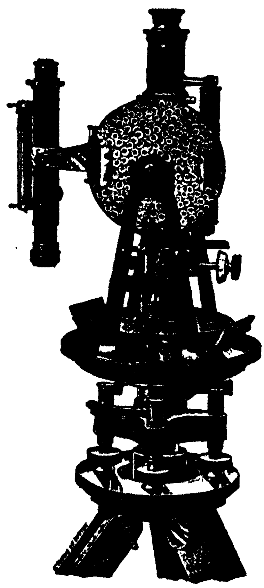
* *Trans. Tech. Soc. of the Pac. Coast*, vol. vii., No. 5.

his telescopic solar attachment (Fig. 45), which Prof. J. B. Johnson says "is accurate beyond any disk-attachment, and represents the correct solution of the solar-attachment problem." The standards, in which the solar telescope may be elevated or depressed, are free to revolve about the polar axis, and are governed by the usual clamp-and-tangent movement. The polar axis is adjusted by trivets at its base-mountings to be at right angles to the line of collimation and transverse axis of the main telescope. When the transit and attachment are in perfect adjustment, its operations are quite precise. The objections to it are mainly the possible errors of the observer in allowing for declination, latitude and refraction; but very careful manipulation will reduce the azimuth of the meridian determined with that of the astronomical North to a few seconds. The original attachment was light in weight and of low telescopic power; but when American mine-surveyors began to employ it in place of the top-telescope its size was considerably increased, and the power of the telescope raised to 18 diameters, as shown in Fig. 45. As used for this double purpose it must be looked upon as the first top-auxiliary, the adjustment of which, for parallelism by means of the base-trivets and for alignment by means of the clamp-and-tangent movement, could be tested and secured by the operator. For vertical sighting it has one inconsiderable fault, to be discussed later, which, in comparison with its other admirable features, cannot be justly said to militate against its successful operation. The mining-transit shown in Fig. 45, having 4-inch circles, graduated to read minutes, and provided with interchangeable eye-pieces to regulate power and light to suit the conditions, must be regarded as one of the best examples of perfect instrumental construction.

FIG. 45.

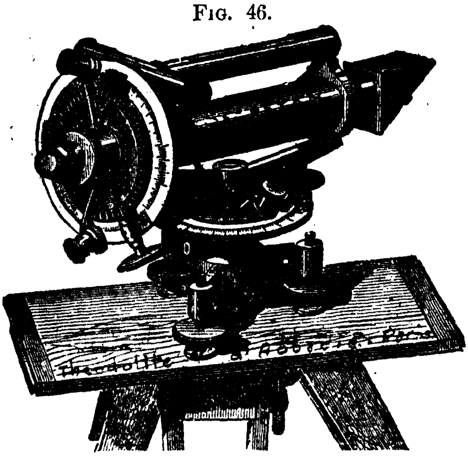


Saegmuller's Telescopic Solar, in Use as a Top-Auxiliary.

In 1885, to meet the ever-prevailing demand for increased efficiency combined with smaller weight, Saegmuller introduced

in America the plain detachable object-prism, which was used for a time for vertical sighting, in place of auxiliary telescopes, but was found to be more alluring than satisfactory. When it was simply fitted to the object-glass, like a sun-shade, the 45° mirror it contained reflected rays truly at right angles to the line of sight, but whether or not in absolute verticality was sheer assumption, and the slightest variation from the correct position doubled the deviation of the emergent rays.

FIG. 46.



Abbadie's Reflecting Theodolite.

This was remedied to some extent by setting a small pillar into the collar of the objective, and so attaching small opposing screws to the prism that they could be made to work upon the pillar. In this way the position of the prism could be regulated very carefully; but to secure for it absolute verticality was an operation that involved more time and trouble than most engineers are willing to take.

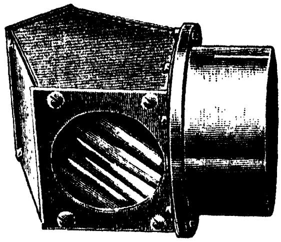
Prof. Steinheil, of Munich, first used, in 1847, for astronomical observations, the object-prism attached to a meridian instrument. Later, M. d'Abbadie, of Paris, employed it, rigidly attached to a small traveler's theodolite (Fig. 46), constructed like a Y-level, mounted upon a circle. The vertical limb of his instrument encircled the telescope at the eye-end. In this way the zenith or nadir position of the prism was determined by bringing the vertical circle to read 0° or 180° upon its vernier.

Before the objective prism came into use, rays were deflected by means of an ordinary mirror, held in the hand, or by attaching to the objective collar a mirror-plate, movable upon a hinge, so as to be set at any angle. In this way Borchers conducted shaft-surveys in 1844 at Clausthal; but the observation of an object at any but a right angle made the calculations complicated.

Last year (1897) Saegmuller brought to perfection the con-

struction of the objective prism. His design is intended to utilize the optical law that *a ray which has been reflected twice in the same plane makes, after its second reflection, an angle with its original direction equal to twice the angle made by the reflecting surfaces with each other.* The double 45° reflecting-prism (Fig. 47), then, will project rays in true verticality when the transit-telescope is placed horizontally, whether the prism be fitted in exact adjustment or not. Therefore the work of this prism in transmitting vertical sights will be as perfect as it is possible to make the adjustment of the telescope to horizontality. As used on American instruments, the objective prism now has but one disadvantage. It is obvious that any necessary movement of the object-glass in focussing destroys the line of sight, and renders variable what should be a fixed eccentricity of collateral sighting. Its most successful operation, therefore, is only with the German or French instruments, which are focussed by movements of the ocular.

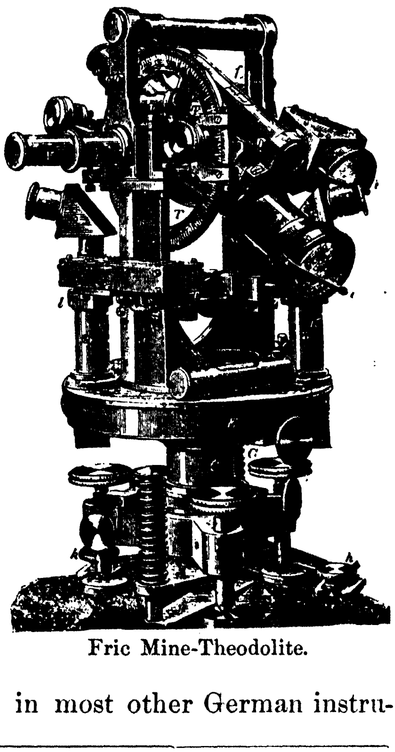
FIG 47.



Double-reflecting Objective Prism.

The most remarkable of modern applications of the prism to mine-transits abroad is exemplified in the instrument (Fig. 48) introduced by Fric Brothers, of Prague, Bohemia, in 1886.* It is claimed by the makers to be an improvement over all other types, to reduce the errors of eccentricity to a minimum, and to overcome the cumbersome features prevalent in most other German instru-

FIG. 48.

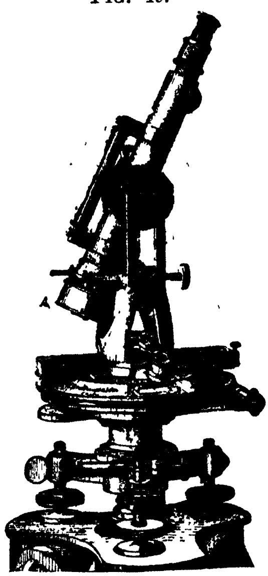


Fric Mine-Theodolite.

* *Zeitschrift für Instrumentenkunde,* Berlin, vi., 221, 1886.

ments. The transverse axis of the main telescope is enlarged and perforated in a manner similar to Hohnbaum's *Grosses Niveau*,* so as to become a secondary telescope for steep sighting. At its outer extremity is rigidly attached a prism, intermediate between the objective and ocular, after the fashion of the "broken telescope" of Reichenbach. It is focussed upon distant objects by a sliding movement of the ocular, as in the mariners' spy-glass, and the barrel is constructed to taper towards the eye-piece according to the scientific principle of the convergence of the rays of light in passing through a lens. The sliding ocular contains the diaphragm and cross-hairs, which are protected from moisture on each side by hermetically sealed thin glass disks; and if by chance any dust-particles should settle upon them, no difficulty is experienced, as the ocular is not focussed upon the plane in which they lie. Spider-webs are hygrometric, being sensibly affected by the humidity of the atmosphere, to the extent of deranging the line of collimation. For this reason Fric suggests that, except for the collection of dust and dew, the occasional German practice of using for the diaphragm a thin glass disk, with delicately etched cross-lines upon its surface, should take the place of spider-webs entirely. The main telescope has a focal length of 17 cm., and is provided with a longitudinal bubble, clamped to its upper surface, that must be removed whenever the telescope is to describe a complete revolution. The horizontal circle is made of thick plate-glass. Its graduations are etched somewhat back from its outer edge, and read by means of the Hensold prismatic glass micrometer with the assistance of reflected light from the prism (*s*) beneath. The needle is placed in a box (*M*) outside the standards, to economize space and weight. It is mounted eccentrically at $\frac{1}{8}$th of

FIG. 49.



Breithaupt's Orientation-Instrument.

* *Die Geometrischen Instrumente*, Hunäus, p. 408.

its length from one end, and balanced by a small counter-weight, so as to give the greatest sensitiveness within the available space.

We notice here another evidence of the decline in the use of the magnetic needle, though for the work of orientation a very precise instrument was introduced by Breithaupt in 1887* (Fig. 49). The needle in this case, which is of more than ordinary length and sensitiveness, is designed to take the place of the magnetometer of Borchers (1846), which was suspended by a silken thread before the objective of the telescope and used to determine exactly the magnetic meridian in mines. The most remarkable of such magnetic surveys known to the writer was the extension of the Ernst-August adit-level in the upper Harz,† through a space of 4753 yards, with a final error of only 8 inches in elevation and 1 minute 8 seconds in azimuth. Owing to dense forests between the shafts, their relative positions were deduced from the Ordnance survey in 1876. This is probably the first instance in which any government survey has served as the basis for important underground work.

The orientation-instrument can be provided with a vertical circle to adapt it to a wider range of work, or, as made by Tesdorpf, with a side-auxiliary telescope, counterbalanced by a bracket-lamp. As originally built, however, it was intended only as a supplementary instrument.

The 15-cm. needle is mounted upon a ruby concentrically with the horizontal circle, and the meridian line of the base is in the same plane with the line of collimation. By use of the additional objective-lens (A) the telescope is transformed to a microscope, and in this way used for the very precise viewing of the needle. This makes it a superior instrument to those provided with the usual form of striding-compass, the concentricity of which with the instrument is not always reliable. The striding-compass, as applied to mine-instruments, was first used in Germany in 1837; but its original inventor is not known. Brander describes them as early as 1780, in connection with sun-dials.

In 1889 Buff & Berger introduced it in America with

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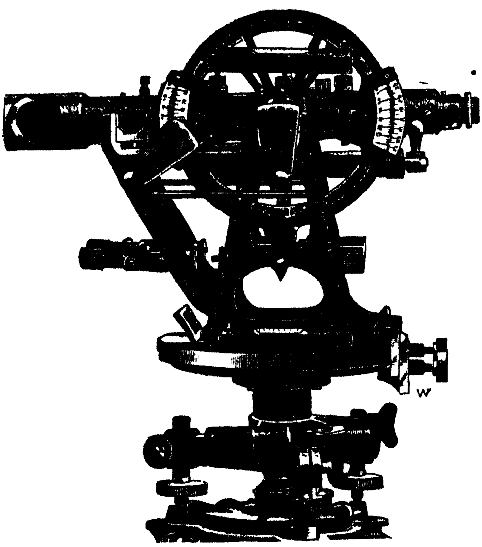
* *Der Bergbau*, No. 24, 1888; also *Cours de Topographie*, A. Habets, Liege, 1895.

† *Berg- und Hüttenm. Zeit.*, li., 293, Aug. 12, 1892.

their duplex-bearing mine-transit (Fig 50), as made for George W. Robinson, of Marysville, Mont. In this instrument vertical sighting could be accomplished, with the main telescope in a position that corresponded to the inclined standards of Young, by removing the telescope, with all its adjuncts, from its normal to its secondary bearings, which were very carefully constructed, being, in fact, cast into one piece with the standards. When in this position, a 4-pound counterpoise was attached to the plates at W.

Later, Keuffel & Esser, of New York, and Fauth & Co., of Washington, each designed instruments of this class, as illus-

FIG. 50.



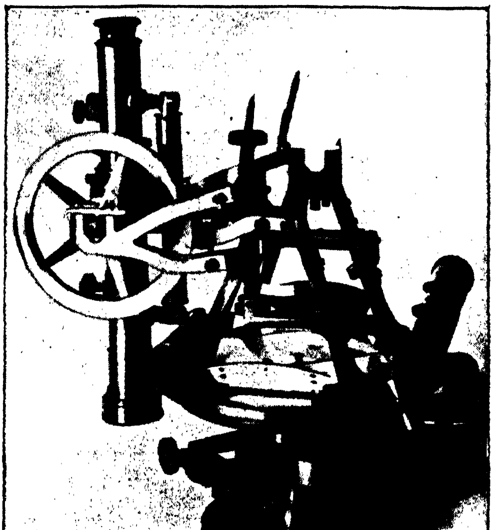
Buff & Berger's Duplex-Bearing Mine-Transit.

trated in Figs. 51 and 52, but have found that their constructions violated the principles which ought to be observed in a perfect transit-instrument. Such conveniences as they afford are hardly commensurate with the risk of getting the delicately-adjusted bearings full of grit underground while making the transposition. "Besides," says Saegmuller, "the instrument was too heavy and too expensive." His instrument weighed 25 pounds complete. The secondary bearings were not permanently fixed to the instrument, but contained in side-arms that were attached, when necessary, by thumb-screws to the

upper part of the standards. To this instrument he added his quick-leveling attachment, patented in 1879. It consisted of two wedge-shaped disks, traveling upon each other in a groove, and interposed between the plates and tripod-head.

This, with the quick-leveling heads of Gurley (1878), constructed upon the principles that obtain in the designs of Pastorelli and Hoffman-Harden, and the detachable ball-and-socket quick-leveling head of Buff & Berger (1883), represent the American achievements along this line.

FIG. 51.

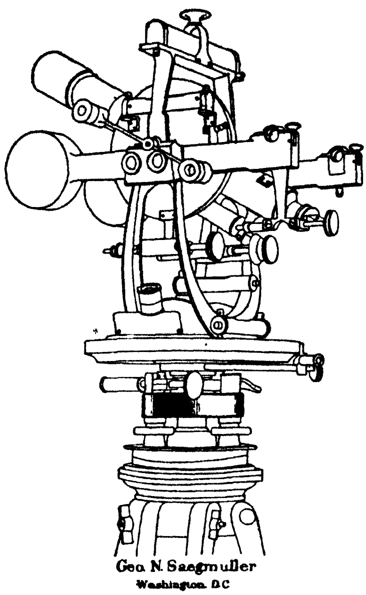


Duplex-Bearing Mine-Transit, Keuffel & Esser.

The concentric model of mine-instrument with the American supplementary telescope must eventually supersede the types of Borchers and Combes abroad; but the popularity of the eccentric instrument in Germany still continues. That recently made by Ludwig Tesdorpf, of Stuttgart (Fig. 53), is one of the few provided with micrometer-microscopes and detachable vertical circle. It has a 12-cm. horizontal circle, reading to 10″, and a 10-cm. vertical circle reading, by vernier, to 1′.

Between the standards is a circular-box bubble for leveling the instrument. The micrometer-microscope is practically the original design of Troughton, in which, briefly, the distance between any degree-line and the index of the limb is carefully measured by a sliding scale that passes through the mutual focus of the objective and the ocular. This is operated by a milled-head screw, of such fineness that one revolution corresponds to 10′ of arc. Each of the sixty subdivisions of the graduated head, then, will represent 10″. The instrument shown in Fig. 53 weighs 6 kg., and its tripod as much more.

FIG. 52.



Duplex-Bearing Mine-Transit, Fauth & Co.

For rapid and accurate subtense measurement some engineers have for a long time been at work on adapting such micrometrical slides to the ocular of the main telescope; but the results of experiments made in the great Indian survey would seem to confine its use still to the determination of the odd seconds in reading horizontal angles.

Prof. Brathuhn has, however, utilized an immovable scale in the diaphragm of the main telescope, on a slightly different principle, that is intended to unite the methods of Dr. Schmidt (p. 712) and the earlier practice of suspending shaft-plumbs in oil and guessing at the probable point of rest.*

The diaphragm he uses is a glass plate upon which the horizontal line is graduated into subdivisions of such value that readings as close as one minute can be estimated. He does not attempt the laborious task of ranging the instrument into exact alignment with the plumb-wires, but sets up at some arbitrary and convenient station, and, by watching the vibrations of each wire upon the scale, determines its angular position

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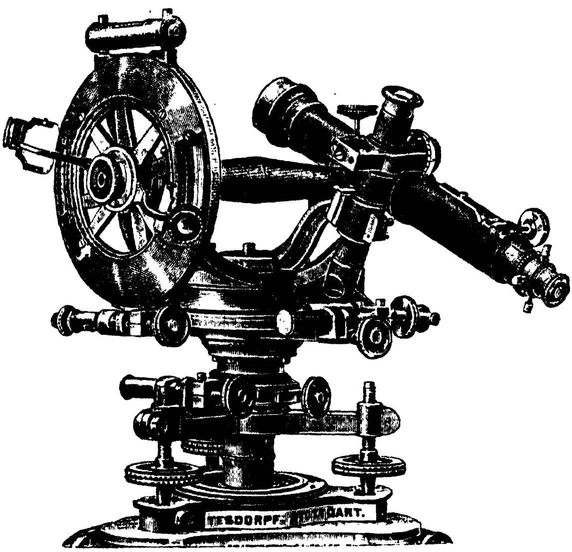
* *Berg- und Hüttenm. Zeitung*, lvi., 395, Nov. 19, 1897.

with reference to the next regularly established station in the survey. The difference of these two readings will give the value of the very acute angle subtended at the instrument from the short base-line between the wires.

One of the most interesting of modern German mine-theodolites is the American pattern of Breithaupt, introduced in 1892 (Fig. 54). It is an improvement upon the generality of American instruments in that the truss-standards (since 1880) have been cast in one piece with the compass-ring; and it possesses also, in the construction of the side-telescope, features well worthy of special comment.

FIG. 53.

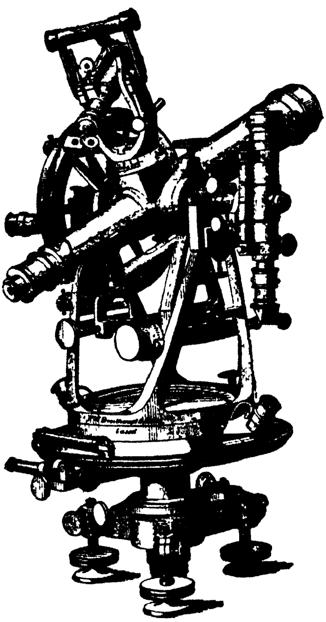


German Eccentric Theodolite.

Its spindle-hub is set into a perforation of the transverse axis in very much the same manner as was customary in the first American model, but has a clamping- and tangent-device that holds it securely, and quickly ranges it into alignment with the main telescope, where its position is verified by a longitudinal bubble. The illumination is accomplished through the transverse axis, as was first practised by Usser, professor of astronomy, at Dublin, in 1790. The 20-cm. horizontal circle is graduated to read by its verniers, or *nonius*, as they are still erroneously called by the Germans, to 10″. Pedro Nuñez, a

Portuguese mathematician, to whom this compliment is paid, published in *De Crepusculis Olyssipone*, 1542, a proposal that upon the plane of the quadrant be described 44 concentric arcs, divided respectively into from 44 to 89 equal parts. The single indicator, then employed, would coincide more or less perfectly with one of the subdivisions. It gave, no doubt, very close readings. For instance, if the index cut the 7th circle at its 43d graduation, the angle was read as $\frac{43}{84}$ of 90°, or 46° 4′ $17\frac{1}{7}$″.

FIG. 54.



Breithaupt's Mine-Theodolite. (American Pattern.)

The solar on the Breithaupt instrument, while practically the design of Saegmuller, is one introduced by Prof. Dr. Schmidt in 1892, for the use of American students at Freiberg.*

Certain engineers took occasion to point out the possibility that Saegmuller's solar might move in altitude upon its horizontal axis while in use for vertical sighting, and thus destroy the efficiency of the base-trivets. In 1895, Buff & Berger made for George T. Wickes, of Cokedale, Mont., an instrument (Fig. 55) calculated to overcome this objection by making the "polar axis" or vertical pillar rigid with the auxiliary telescope, retaining only the trivet-base, in a new form, with every desirable provision to insure parallelism and alignment. As in this construction the setting of the sun's declination becomes impossible, its uses are restricted to the primary offices of a top-auxiliary telescope; but as such, with its delicate, rapid and effectual means for all necessary adjustments, it represents, no doubt, with Breithaupt's side-auxiliary, all that could be required in such individual devices.

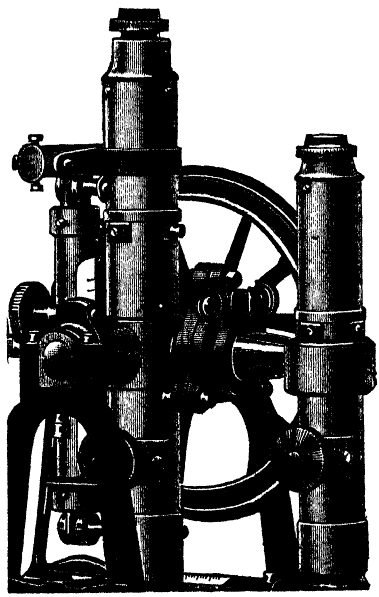
No matter how perfect may be the construction and means of adjustment, however, each of these appliances has, combined

* *Oesterr. Zeit. für Berg- und Hüttenwesen*, No. 21, 1892.

with its advantages, that negative condition known as eccentricity, for which correction must be allowed, varying with the conditions in each case.

In 1896 the writer designed a mine-tachymeter which, as he ventures to assert, by its peculiar yet simple construction, embraces the advantages and eliminates the disadvantages of all other types. Its individuality consists principally in the *interchangeability* of the auxiliary telescope and the means provided to thus transform the instrument from one condition to the other, as shown in Figs. 56 and 57.

FIG. 55.



Buff & Berger's Top-Telescope, with Adjusting Trivets.

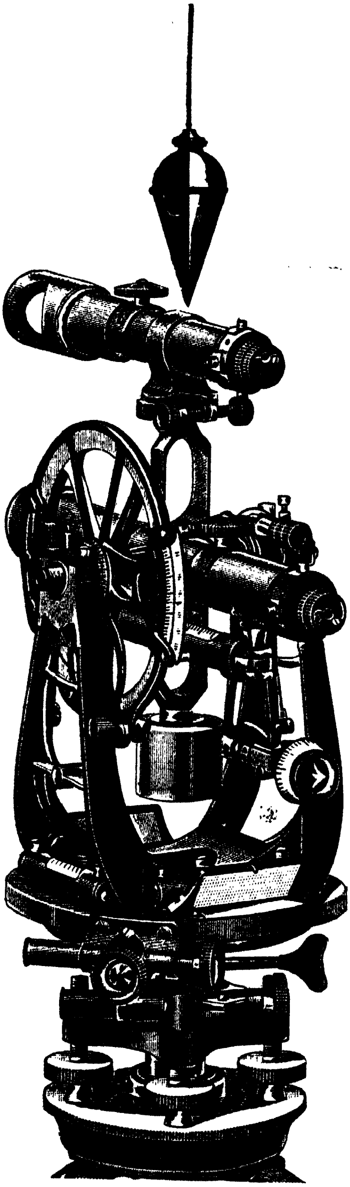
In this way the double negative quantities become positive in their resultant, so to speak; and we have a mining-transit capable of performing, with more than usual exactness, all the complex functions required in mines, and requiring *absolutely no corrections for eccentricity.*

The auxiliary telescope is so provided with a hub of new design that it may be screwed to the threaded extension of either the transverse axis or the vertical pillars of the main telescope. In this position it is clamped firmly and ranged quickly into alignment with the main telescope by two small opposing screws that work up an arm of the hub. Upon its diaphragm is but one web, so placed that it shall be vertical when on top, and horizontal when at the side. In either position the amount of eccentricity is the same, though perfect operation would not be affected if this varied, since the observation of steep horizontal angles is made only with the auxiliary on top, and of very precipitous vertical angles with the auxiliary at the side.

On this account, any adjustment for parallelism in the optical axes of both telescopes is dispensed with, as its peculiar

adaptability will insure perfect results even if the conditions in this particular are imperfect.

FIG. 56.



Scott's Mine-Tachymeter. Auxiliary on Top.

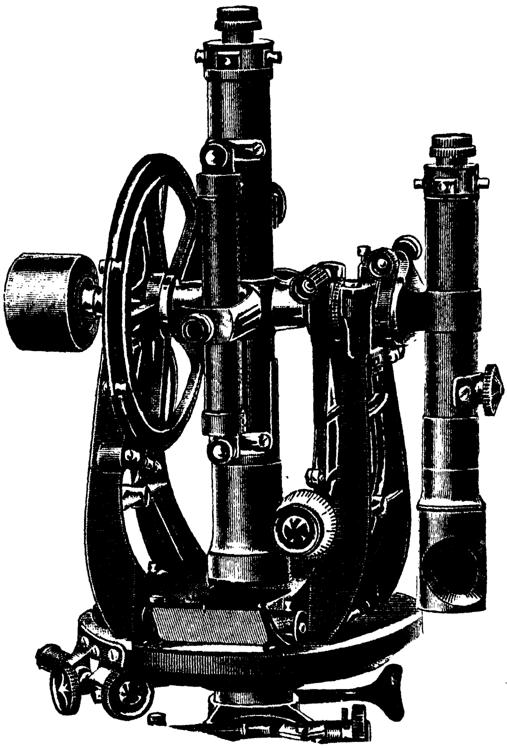
Buff & Berger, who made the instrument, have recently added trivets to the base of the upper vertical pillar; but these are unnecessary, and impair the stability of the instrument.

The auxiliary (Fig. 58) has a power of 17 and the main telescope of 24 diameters, being the greatest possible under the restrictions observed with regard to size and light. The amount of light received through the ocular varies as the square of the diameter of the objective; therefore, the larger the aperture in mine-transits the more favorable will be the conditions with respect to light, provided, however, that power be not sacrificed by the use of an ill-proportioned ocular. The ocular of this instrument is inverting, conforming to the general practice of European engineers, who no doubt excel in this respect. As American engineers become better acquainted with their desirable qualities, either the Ramsden, Kelner or Steinheil oculars will be more widely used. They all have the advantage of not only permitting greater light and a larger field, but in a telescope of the same size an objective of greater focal length is permissible, thereby favoring the conditions imposed to secure the best definition. By thus increasing the focal length of the objective, while, by virtue of its construc-

tion, that of the inverting ocular is decreased, the magnifying power becomes greater.

Both horizontal and vertical circles are 5 inches in diameter, divided into half degrees, and read to minutes. This, on the authority of many years' practice, and by general consent, is conceded to be most easily read underground, and to be fine enough for mine-work. The novice is generally too much inclined to high telescopic power and extremely fine graduations,

FIG. 57.



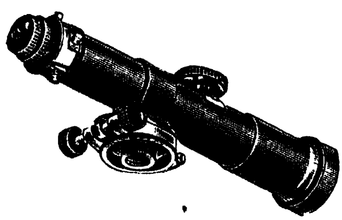
Scott's Mine-Tachymeter. Auxiliary at the Side.

with the idea that the greatest accuracy can thus be attained. But this is a mistake. Beside, the stationary double-lens reading-glasses that become necessary to read such fine circles only provide the means of invariably burning the engineer's face when he attempts to get both his eye and candle near enough to the plates to take a correct reading. The vertical circle is graduated in quadrants, the zero-line running parallel with the line of collimation, and is read by one double vernier, so placed

near the eye-end that any angle of elevation or depression may be determined with the telescope in a normal or reversed position. The horizontal graduations read only in one direction, being numbered continuously with one set of figures, from 0° to 360°. This permits the verniers to be single, and provides a uniform method that almost entirely removes any possibility of error in reading, recording, figuring or platting. The verniers are directly under the telescope, so that one need not move to read them, and if the engineer is satisfied that his graduations are correct, he will habitually read but one of these, taking care, however, to repeat every angle at least once; for no mine-surveyor can be certain of his work until he has checked every step by the same or different means.

The U-shaped standards are a new pattern, designed to conform to American practices and methods. Being of one piece they are very rigid, and, as old-time fancies wear out, will doubtless come into general use. They are made of aluminum, and bushed with electrum at the bearings. Their construction does not permit the use of the usual compass-box; but in high-class mine-work the magnetic-needle cannot seriously be said to be essential for any purpose whatever. The history of magnetic surveys is itself the death-warrant of the miners' compass; and in this age of widespread electrical power and lighting (employed with rapidly increasing frequency in mines), the magnetic needle becomes no more reliable in mine-surveys than on the present iron-clad man-of-war.

FIG. 58.



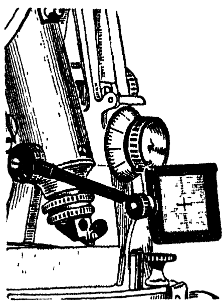
Interchangeable Auxiliary of Mine-Tachymeter.

Mine-surveys are nearly always figured by trigonometrical functions, as referred to the boundary-lines; but if the engineer prefers to use the calculation by latitudes and departures, a good practice is to establish by stellar or solar observation, at one end of the base-line in the surface-triangulation, or, better, at one of the boundary-corners, a true meridian from which every station in the whole system, both underground and on the surface, has an established latitude and departure, and every course an established bearing. After the work of procuring

and tabulating these data is once completed, this system is perhaps the most concise in subsequent computation; but the initial time and effort it requires are scarcely repaid by the benefit secured.

For such work, with this tachymeter, the Davis solar screen (Fig. 59) is doubtless best to use. It was invented in 1880 by Prof. J. B. Davis, of the University of Michigan, who, in writing to me, said: "In my opinion, my solar screen requires less calculation than any other, if properly used. Its work is even more precise than the circles of the transit, and requires no special adjustments or mechanical conditions. Some others require those that cannot be tested." If used with an erecting telescope, the full aperture of the objective is utilized; but with an inverting ocular, in order to obtain a clear reflection of the cross-hairs upon the screen, a telescope-cap is provided, so as to reduce the aperture to about ¼-inch.

FIG. 59.



Davis Solar Screen.

The diaphragm of this instrument is made of more than ordinary thickness. Upon one side are placed the usual cross-hairs and upon the other the fixed stadia-hairs, which are out of focus when not in use. In this way, as explained before (p. 720), there will be no danger of reading an important vertical angle on a long and indistinct sight with the wrong horizontal hair.

The shortest sight possible with the telescope of this mine-tachymeter is 5.5 feet, which for ordinary mine-work is sufficient, though occasionally a shorter sight than this is unavoidable. In most German and French instruments the ocular can be drawn out so far as to permit observations within the first meter; but this plan is impossible in American and English models. For these, then, the only plausible plan for very near sighting must provide for an additional objective lens, as described in connection with Breithaupt's orientation instrument (see A, Fig. 49). Such an arrangement Buff & Berger are now perfecting for this work. It is provided with adjusting-screws at the side, so that the center of the lens may be made to coincide exactly with the optical axis of the telescope.

Otherwise, by the careless addition of an extra objective, the adjustment of the line of collimation may be disturbed.

In designing the mine-tachymeter, it was the writer's object to

Fig. 60.

The First American Transit, built by Young & Son, Philadelphia, 1831.

make it the most complete, convenient, precise and compact instrument yet introduced for mining engineering, and to this end it was his intention to add one other improvement, which

up to this time remains but a suggestion; but such engineers or makers as choose to employ it may do so without fear of interference, as the original makers are now seemingly extinct.

In the *Eng. and Min. Journal* of November 7, 1891, is described Cook's patent luminous level tube, which has an inner coating of phosphorescent compound, covered by a coat of water-proof lacquer, by which the bubble is made to appear as distinct against the graduations in the tube in the dark as in the light. As it frequently happens that, because the flicker of surrounding lights seems to absorb all dim rays coming from a long, indistinct sight, the engineer prefers to remain in the dark, the use of such a device would enable him to watch his bubbles while making such observations.

In ordinary setting-up, moreover, it seems likely that the work would be greatly facilitated.

Fig. 60, a picture of the first American transit, referred to on p. 703, may fitly conclude this paper, showing how much progress has been made in the construction of such instruments since that modest beginning, sixty-seven years ago.

SECRETARY'S NOTE.—An interesting and valuable discussion of this paper will be issued in pamphlet form and included in the next volume of the *Transactions*. Both paper and discussion will also be published by the Institute in a separate volume.

Note on the Possible Origin of the Pneumatic Process of Making Steel.

BY WILLIAM B. PHILLIPS, PITTSBURGH, PA.

(Buffalo Meeting, October, 1898.)

IN connection with the address of our late President, Mr. Joseph D. Weeks, delivered at the Pittsburgh meeting, in February, 1896,* I venture to believe that a circumstance which came recently to my notice may possess some historical interest.

Several years ago I spent some months in Western Kentucky

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* *Trans.*, xxvi., 980.

near the site of the old Kelly furnace, in the vicinity of Kuttawa. I became acquainted with many men who had known William Kelly well. Some of them had worked for him, and had very lively recollections of "Old Phosphorus," as he was called.

Somewhere about 1845, when he was working on the pneumatic process which afterwards bore his name, and was used by W. F. Durfee at the old Wyandotte works, he imported four Chinamen. He secured them through the American consul in China, and they worked at his iron-furnace. Now, there is an old story that the Chinese had refined iron by blowing air into it a great many years ago, and I have thought that Kelly, in asking for Chinese laborers, would naturally require the services of those who had some knowledge of the iron business. He would have had no use for the ordinary Chinese laborer, since there was, in the region where he operated, no special scarcity of labor of the unskilled sort. May it not be that he was aided by his Chinamen in experimenting with the pneumatic process, and that some knowledge of this method of refining iron was brought into this country first by those men? This is merely a suggestion, and I am unable to say whether it is worth following up.

The International Correspondence Schools, Scranton, Pa., With Special Reference to the Courses in Mining.

BY H. H. STOEK, SCRANTON, PA.

(Buffalo Meeting, October, 1898.)

AMONG the mining and metallurgical achievements of the latter part of the nineteenth century, not the least is the inception and successful prosecution by mining men of a technical educational movement which in magnitude surpasses any similar movement ever before set on foot. The following account of this undertaking has been prepared at the request of the Secretary of the Institute:*

* SECRETARY'S NOTE.—To the preliminary pamphlet edition of this paper I added a note, intended to relieve Prof. Stoek, the present Director of the Scranton schools, from any appearance of having volunteered what might be deemed an

History and Scope.

Instruction by correspondence is by no means a new thing; but within the past 25 years a marked impetus has been given to such instruction, as is evidenced by the numerous Chautauquan courses and schools which have sprung up throughout this and other countries.

The International Correspondence Schools, while a part of this general movement, are not, as might naturally be supposed, a direct result and outcome of it. They trace their origin to a direct demand by the miners for assistance in preparing themselves for the examinations for positions as foremen, etc., necessitated by the educational qualifications imposed upon applicants for such positions by the mining laws of the several States enacted during the past 15 years.

Pennsylvania has not only been one of our foremost producers of mineral wealth, but on account of the unusual dangers connected with coal-mining she has always been foremost in mining legislation, and was the first State to impose, in 1885, an educational qualification upon applicants for positions of responsibility.

The *Colliery Engineer*, of Scranton, Pa., heartily supported this measure, and, after its enactment, rendered great assistance to the miners of Pennsylvania through its "correspondence" or "answers to questions" department in preparing them for the State examinations provided by the new law. Interested persons were urged to ask questions or to answer those asked by others upon any subject pertaining to mining; and this feature of the paper was so greatly appreciated, and so numerous were the responses, that it was no unusual thing to receive for publication in each issue more than enough matter to fill the entire paper. It soon became apparent that it was impossible to supply through this medium alone the instruction and assistance sought for; and in August, 1891, the Colliery

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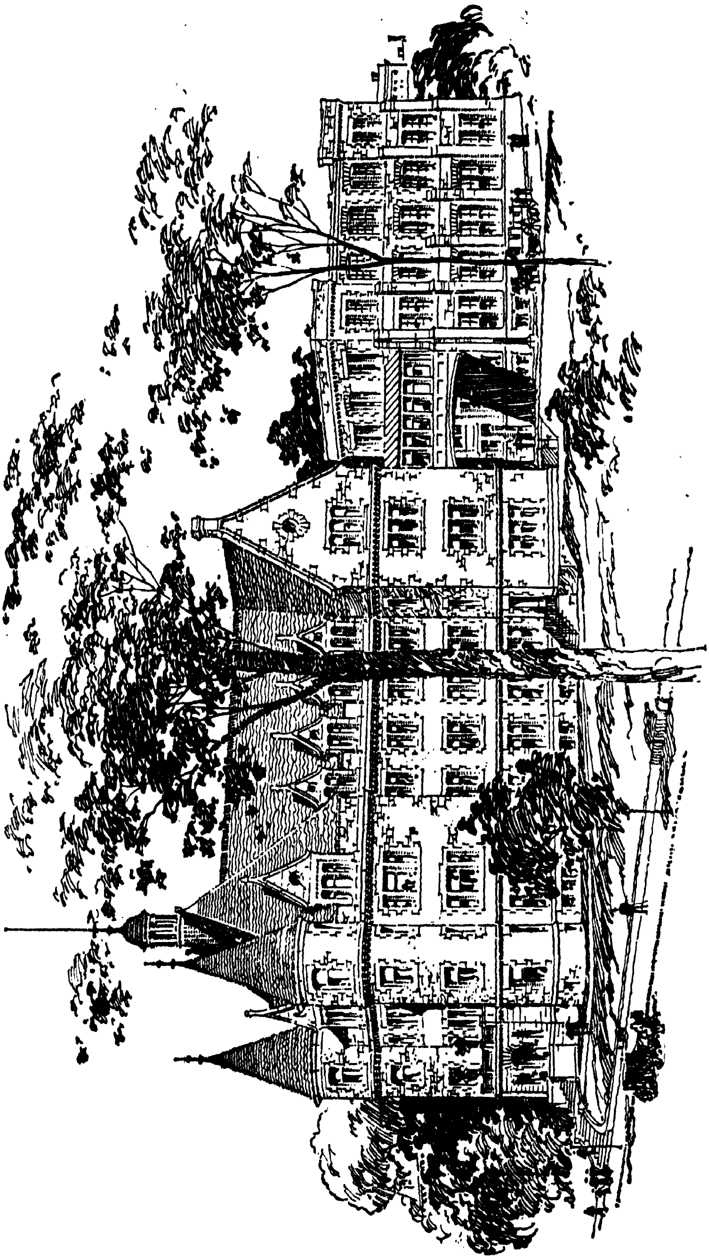
advertisement of a private business enterprise. In my judgment, the importance of the subject outweighed this possible objection.

A paper on "Correspondence-Schools," presented at the New York meeting, February, 1899, by Mr. R. P. Rothwell, New York City, which has been distributed to members in pamphlet form, and will be published in the next volume of the *Transactions*, gives a more comprehensive view of the subject, and enumerates other enterprises of similar character.—R. W. R.

Engineer Co. began the preparation of a course covering the subjects of coal-mining, mine-surveying, mine-machinery, etc., which developed later into the complete coal-mining course. In accordance with the results of a wide journalistic experience and 25 years' acquaintance among mining men, acquired in conducting the *Mining Herald* and the *Colliery Engineer*, it was decided, in order that the course might be open to any one, no matter how limited his earlier advantages might have been, to commence with arithmetic and to advance gradually, step by step, to the higher branches. Instead of attempting to use the ordinary text-books, which are usually prepared for collegiate instruction and assume considerable preliminary preparation, a series of instruction-papers was prepared embodying, in the simplest manner possible, the principles of the several subjects. These papers form a characteristic and distinct feature of this system of instruction, and the results obtained with the first ones issued were so satisfactory that the same method has been pursued in the preparation of the subsequent courses. The complete coal-mining course became popular at once. It had been estimated that perhaps 300 students would be enrolled during the first year; but so hearty was the reception given to the plan that, at the end of its first year, 1200 had been enrolled in the course, and the practicability of the method employed had been demonstrated.

During the first year, in response to many inquiries received from mine-machinists, stationary engineers, pump-runners, etc., a course was started called the "mine-mechanical," which included the mechanical papers of the complete coal-mining course, with the addition of mechanical drawing and allied subjects. About this time, also, prospectors, assayers and metal-miners requested instruction in blow-piping, mineralogy and prospecting; and the "metal-prospectors'" course was begun, out of which, by similar processes of differentiation, the "metal-mining" and "full mining courses" have been developed. Early in 1892 the preparation of the "complete mechanical" course was commenced, and within 18 months after its establishment over 2000 enrollments were made in this course alone. Since that time the demand for other courses has continued; and at the present time 65 distinct courses are offered, divided among 12 different schools, as follows:

FIG. 1.



Buildings of the International Correspondence Schools, Scranton, Pa.

The Correspondence School of Mines.

Full Mining; Complete Coal-Mining; Metal-Mining; Mine-Mechanical; Short Coal-Mining; Metal-Prospectors'; Electric Mining (classed under the School of Electricity).

The Correspondence School of Chemistry.

Chemistry, including Qualitative and Quantitative Analysis; Chemistry, including Qualitative Analysis; Inorganic and Organic Chemistry.

The Correspondence School of Steam-Engineering.

Stationary Engineers'; Marine Engineers'; Locomotive-Engineers'; Traction-Engineers'; Gas-Engineers'; Refrigeration.

The Correspondence School of Electricity.

Electrical Engineering; Electrical; Electric Power and Lighting; Electric Lighting; Electric Railway; Electric Mining; Wiring and Bell-Work; Telegraphy; Telephony.

The Correspondence School of Civil Engineering.

Civil Engineering; Railroad-Engineering; Surveying and Mapping; Bridge-Engineering; Municipal Engineering; Hydraulic Engineering.

The Correspondence School of Mechanics.

Complete Mechanical; Mechanical Drawing.

The Correspondence School of Architecture.

Complete Architectural; Architectural Drawing and Designing; Architectural Drawing.

The Correspondence School of Plumbing, Heating, and Ventilation.

Sanitary Plumbing, Heating, and Ventilation; Sanitary Plumbing and Gas-Fitting; Sanitary Plumbing; Gas-Fitting; Heating and Ventilation.

The Correspondence School of Sheet-Metal Pattern-Drafting.

Sheet-Metal Pattern-Drafting; Tinsmiths' Pattern-Cutting.

The Correspondence School of English Branches.

English Branches; Arithmetic; Spelling; Penmanship; Grammar; Letter-Writing; Geography; United States History; United States Civil Government.

The Correspondence School of Bookkeeping and Stenography.

Complete Commercial; Complete Stenographic; Bookkeeping and Business Forms; Single Entry Bookkeeping; Double Entry Bookkeeping; Opening, Closing, and Changing Books; Stenography.

The Correspondence School of Pedagogy.

Pedagogics of English Branches; Pedagogics of Arithmetic; Pedagogics of Grammar; Pedagogics of History; Pedagogics of Orthography; Pedagogics of Geography.

The subjects included in the mining courses are as follows:

The Complete Coal-Mining Scholarship.—Arithmetic; formulas; geometry and trigonometry; gases met with in coal-mines; mine-ventilation; geometrical drawing; mine-surveying and mapping; economic geology of coal; prospecting for coal; shafts, slopes, and drifts; methods of working coal-mines; mechanics; steam and steam-boilers; steam-engines; air and air-compression; hydromechanics and pumping; haulage; hoisting and hoisting-appliances; surface-arrangements of bituminous-mines; surface-arrangements of anthracite-mines; compressed-air coal-cutting machinery; percussive and rotary boring.

The Mine-Mechanical Scholarship.—Arithmetic; mensuration and the use of letters in algebraic formulas; mechanics; geometrical drawing; mechanical drawing; steam and steam-engines; steam-boilers; air and air-compression; hydromechanics and pumping; haulage; hoisting and hoisting-appliances; surface-arrangements of bituminous-mines; surface-arrangements of anthracite-mines; compressed-air coal-cutting machinery; percussive and rotary boring; dynamos and motors; electric haulage and hoisting; electric pumping, lighting, and signaling; electric coal-cutting machinery.

The Metal-Mining Scholarship.—Arithmetic; formulas; geometry and trigonometry; geometrical drawing; mine-surveying and mapping; blow-piping; mineralogy; assaying; economic geology; prospecting for gold and silver; placer and hydraulic mining; preliminary openings; permanent openings; methods of working metal-mines; gold- and silver-milling machinery; mechanics; steam and steam-boilers; steam-engines; air and air-compression; hydromechanics and pumping; haul-

age; hoisting and hoisting-appliances; percussive and rotary boring.

The Metal-Prospectors' Scholarship.—Blow-piping; mineralogy; assaying; economic geology; prospecting for gold and silver; placer and hydraulic mining.

The Full Mining Course.—Arithmetic; formulas; geometry and trigonometry; gases met with in coal-mines; mine-ventilation; geometrical drawing; mine-surveying and mapping; economic geology of coal; prospecting for coal; shafts, slopes, and drifts; methods of working coal-mines; mechanics; steam and steam-boilers; steam-engines; air and air-compression; hydromechanics and pumping; haulage; hoisting and hoisting-appliances; compressed-air coal-cutting machinery; percussive and rotary boring; surface-arrangements of bituminous-mines; surface-arrangements of anthracite-mines; dynamos and motors; electric haulage and hoisting; electric pumping, lighting, and signaling; electric coal-cutting machinery; blow-piping; mineralogy; assaying; economic geology; prospecting for gold and silver; placer and hydraulic mining; preliminary openings; permanent openings; methods of working metal-mines; gold- and silver-milling machinery.

The Short Coal-Mining Scholarship.—Arithmetic; mensuration and trigonometric functions; gases met with in mines, etc.; mine-ventilation; economic geology of coal; prospecting for coal; shafts, slopes, and drifts; methods of working coal-mines; mine-surveying; mine-machinery.

Electric-Mining Scholarship.—Arithmetic; formulas; mensuration; dynamos and motors; electric pumping, lighting, and signaling; electric haulage and hoisting; electric coal-cutting machinery.

Method of Instruction.

The method of instruction differs materially from that followed by Chautauquan and other correspondence schools. Each subject taught is presented through the medium of one of the instruction-papers, which have been carefully prepared by engineers in active practice, or by those in the employ of the Schools. These instruction-papers are digests and compilations from the text-books and periodical literature of the day, supplemented from the practical experience of the writers. Each one is made as far as possible complete in itself, depending

simply upon what has been given in the preceding papers of the course. Great stress has been laid upon a clear and concise style of presentation, illustrations having been profusely employed as aids in this respect, and it can safely be said that the illustrative part of these papers has not been excelled, if ever equaled, in the presentation of technical subjects. Up to the present time 216 of these papers have been prepared and copyrighted, at a cost for authorship and printing of more than $200,000. The first editions of the papers are limited in number, in order that each may be subject to constant revision, so that it can be kept entirely up to date, and modified in the light of difficulties encountered by the students in the study of it—a record of all these difficulties being preserved and consulted at the time of revision. These instruction-papers are thus seen to possess advantages over text-books in being more easily kept up to date.

Each instruction-paper is accompanied by a question-paper, and, after the former has been studied, the questions upon the latter are answered in writing and mailed to Scranton, where these answers are examined and all corrections are noted in red ink upon the answer-sheet, which is then returned to the student, together with the suggestions of the examiner. Answers not receiving a grade of 90 per cent. must be wholly or partially re-written; and no paper is considered complete until every question upon it has received an answer deserving the mark of 90 per cent. When the student is enrolled in the School, he receives the first two instruction-papers; upon completion of the first of these he receives No. 3; and so on in order, so that he always has one paper to study. The regular order of the course is, however, carried out rigidly; and advance-papers can be obtained only upon the completion of a preliminary one, except in cases where bound volumes of instruction-papers are furnished after a portion of the course has been completed. Upon the completion of the entire course an examination is held upon all the subjects of that course before a certificate of proficiency is issued.

When the student meets with difficulties in the subject he fills out an information-blank and sends it to Scranton; all such requests for information are immediately and fully answered, it being a part of the agreement made with the stu-

dent upon his enrollment that all requests for information will be promptly answered. These requests for information are filed and utilized in revising the instruction-papers as mentioned above.

If the student fails for a considerable time to send in work, it is understood from his silence that he has given up the work for one reason or another, and a letter is written to him by one of the instructors urging him to resume it, and promising him the exclusive service of an instructor, if necessary, to help him to understand it. A record is kept of the instructor who writes this letter to the student, and if the latter answers and resumes his work, all of it is cared for by the "special instructor." In this way many students are encouraged to proceed with their courses who otherwise would not be able to do so. Students and instructors are thus kept in close contact by frequent correspondence, and every effort is made to compensate in this way for the lack of personal intercourse between teacher and pupil.

The instruction-papers become the property of the student, who, however, pledges himself to reserve them strictly for his own use. In addition to the pamphlet edition of the papers, and after completing a portion of the course, he receives the entire set finely printed and substantially bound in half-leather, thus acquiring a set of books which are an ornamental and valuable addition to the library of any engineer. The bound volumes are also issued in limited editions, the plates from which they are printed being subject to revision, as above described, in connection with the account of the instruction-papers.

Cost.—The charges for the course vary from $5 for a course in arithmetic to $165 for a complete course in civil engineering, the charges for the courses in mining being as follows:

	Cash in advance.	$5 per month.	$2 per month.
Full mining,	$60.00	$65.00	$70.00
Complete coal-mining,	42.00	47.00	51.00
Metal-mining,	42.00	47.00	51.00
Mine-mechanical,	41.00	46.00	50.00
Short coal-mining,	29.00	34.00	38.00
Metal-prospectors',	18.00	20.00	23.00
Electric mining,	25.00	30.00	34.00

These prices include a set of the bound volumes and the postage upon all correspondence sent to the students.

With reference to such subjects as mechanical drawing, blow-piping, mineralogy, assaying, etc., which are usually considered to require laboratory-facilities, the question naturally arises, "How can these subjects be taught by correspondence?" A few additional details of the methods employed in these subjects will, therefore, not be out of place.

As with other topics, instruction-papers in mechanical-drawing are provided, describing the character and use of drawing-instruments, methods of penciling and inking drawings, and giving detailed instructions for making each plate. The question-paper used in other subjects is replaced by a plate, which is to be drawn by the student in accordance with the instructions in the instruction-paper and mailed to Scranton for correction and revision. The instructor notes in pencil upon the drawing the points which can be improved, and makes full suggestions for the guidance of the student, and, when necessary, writes a detailed letter for his assistance. The corrected work, with these suggestions, is then returned to the student, and may have to be re-drawn. After he has received a passing mark of 90 upon the plate, a new one is sent him. The process is repeated with from 15 to 40 plates, including several tracings, the number depending upon the course. After an experience of eight years in teaching mechanical drawing in two well-known technical colleges, I have no hesitancy in saying that the results obtained by this method far surpass anything that I thought possible; and the progress made by fire-men, machinists, etc., with no other instruction than that received through these schools, is remarkable.

In blow-piping and mineralogy, in addition to the instruction- and question-papers, an outfit is furnished, containing the usual blow-piping appliances, together with a number of mineral specimens. With this the student is required to work upon the specimens furnished and to report upon them.

In assaying, no attempt is made to give practical instruction. The instruction-papers deal simply with the theory of assaying, and give complete descriptions of all the apparatus used, and the methods of operating the same.

The work of instruction is directly under the supervision of

the Manager and Assistant Manager; and at the head of each school there is a principal, under whom are a number of instructors, who examine and correct the papers under the immediate supervision of the principal.

In addition to the instruction-corps, engineers are also engaged in writing and editing new instruction-papers, and in revising the courses already used, preliminary to having them printed in the bound volumes. In these several capacities there are employed at the present time 50 engineers, most of whom are graduates of American or European technical schools.

SIZE OF SCHOOLS.

At the present time (November 15, 1898) 63,246 students are enrolled in all of the Schools, of which number 5608 are taking the courses in mining—a remarkable development in 6 years. The character of the students will be best shown by the following tabulated statement of the positions occupied by men about the mines who are taking the courses: 11 State mine inspectors; 28 mine operators; 26 secretaries and treasurers of mining companies; 306 superintendents of mines; 34 assistant superintendents of mines; 78 mining engineers; 22 assistant mining engineers; 102 civil engineers; 112 mine-surveyors; 36 mine-surveyors' assistants; 745 mine-foremen; 126 assistant mine-foremen; 189 fire-bosses; 22 mine-contractors; 2512 coal-miners; 17 mining-machine runners; 22 slope-men, foot-men, etc.; 14 slate-pickers; 465 metal-miners; 50 metal-prospectors; 165 mine-laborers; 45 drivers in mines; 36 driver-bosses; 8 loader-bosses; 78 weigh-masters; 53 coal-inspectors; 73 mine-trackmen; 67 mine-carpenters and timbermen; 104 mine-machinists; 31 mine-blacksmiths; 22 coke-foremen; 11 quarry-foremen; 14 mill-foremen and superintendents; 34 mill-men; 14 amalgamators; 11 ore-sorters and samplers; 67 assayers; 42 chemists; 8 metallurgists; 20 mine-brokers and land-commissioners; 73 coal-agents; 272 mine-clerks, stenographers, bookkeepers, accountants, time-keepers, etc., and 2089 in miscellaneous mining occupations.

More than 100 graduates of American and European technical schools are at present enrolled as students, either reviewing the courses in which they have already graduated or studying along new lines.

Management.

The International Correspondence Schools are owned by the Colliery Engineer Co., a stock corporation, capitalized at $1,250,000. The officers of the Company are: Mr. T. J. Foster, Manager; Mr. R. J. Foster, President; Mr. E. H. Lawall, Vice-President, two of whom are members of this Institute. The Board of Trustees consists of these officers, together with Mr. J. K. Griffith, Superintendent of the Latrobe Steel Works, and Mr. Frank T. Patterson, of Philadelphia, a prominent coal operator of the firm of Geo. B. Newton & Co. and a Director of the Fourth Street National Bank, of Philadelphia. Among the stockholders will be found the names of a large number of men prominent in mining and metallurgical matters. The Company owns its own buildings on Wyoming Avenue, Scranton, Pa., erected at a cost of $250,000. These buildings are two in number, the larger, as shown in Fig. 1, being five stories in height and covering a ground-space of 6942 square feet, while the smaller is four stories high and covers 2646 square feet of ground-space. The former of these is devoted to the instruction and financial offices of the Schools, while the latter contains one of the best equipped printing- and binding-plants in the country. So rapid has been the growth of this institution that the present commodious quarters are already overcrowded, and other buildings are contemplated. These buildings were designed by Mr. W. Scott-Collins, Principal of the School of Architecture.

Connected with the Colliery Engineer Co., and devoting their time exclusively to its interest, there are at the present time the following number of employees: Managing Department, 7; Instruction Department, 193; Treasurer's Department, 67; Supply and Shipping Department, 12; Extension Department, 300; Printing Department, 124; Editorial Department, 19. The value of the Schools to the City of Scranton is evidenced by the above list of employees, and by the fact that they pay out each year, for postage alone, over $50,000.

The practicability of the correspondence-method of instruction has been proved past all peradventure by the continued success of these Schools and by the testimonials of those who have taken the courses. This success is due to two things:

(1) The character of the students, who are not school-boys, but men who are spending their own hard-earned savings,

and who have the foundation of practical experience upon which to base the theoretical superstructure.

(2) The extreme care which has been taken in the preparation of the instruction-papers and the very clear manner in which the subjects are therein presented.

The only dissatisfied and complaining students are those who have entered upon the courses failing to comprehend that work would be required of them in order to succeed. The man who thinks his part is done when the money is paid in will be sadly disappointed, for while every inducement is held out, and every possible assistance is rendered, there is no royal short-cut along the correspondence-road any more than along the beaten paths going through the school-room.

It is not claimed that correspondence-instruction is the equivalent of, or can replace, collegiate methods; and when an applicant for enrollment in the Schools has the time and means to take a course in one of our colleges he is invariably advised to do so. But for the larger number of men and women who are unable to go through the more extended course, the International Correspondence Schools offer an opening for self-improvement which cannot be over-estimated.

The Superficial Alteration of Western Australian Ore-Deposits.

BY HERBERT C. HOOVER, COOLGARDIE, WESTERN AUSTRALIA.

(Buffalo Meeting, October, 1898.)

WHILE the importance of the superficial alteration of gold-deposits is recognized in pure science, it seems often to be singularly disregarded among engineers in practice. Its great practical bearing lies in the exploitation of new regions; and nowhere are the disastrous results of ignoring it more vividly shown than in Western Australia, where the impoverishment of the ores, and their frequent transition from free-milling to refractory character in depth (things, as a rule, neither anticipated nor ascertained by exploration beforehand), have brought many companies to ruin, or caused the loss of much money squandered on machinery adapted only to temporary conditions.

The requirements of the market have not allowed to finance companies or engineers proper time to exploit before flotation or equipment; and in many instances there has been a disposition to avoid sinking, and thereby perhaps spoiling a good mine for the market!

The general nature of the phenomena of superficial alteration in gold-deposits is so fully understood as not to need discussion here. This brief paper will, therefore, point out only the special peculiarities of the phenomena observed in Western Australia.

Nature of the Gold-Deposits.

The Western Australian deposits may be placed in two classes: (1) impregnations of the country-rock along fissure-planes or crushed zones, with but little deposition of gangue-minerals; and (2) normal quartz fissure-veins. The surface-phenomena of these classes are widely different, and they will therefore be discussed separately.

Impregnations.—The deposits at Kalgoorlie, and in a few isolated instances in outside fields, such as the "Sons of Gwalia, Ltd.," at Mt. Leonora, fall under this heading. It may be noted here that of the fifteen or sixteen important mines in the Colony, ten are of this character.

These deposits may be briefly described as zones of crushing and fissuring in diabase and highly altered slates, from which channels impregnation and some replacement of the country-rock have taken place.

The dynamic phenomena are strongly evidenced by the frequent walls (which, however, do not represent lateral extensions of the ore), and by the not uncommon brecciation and crushed zones. The lodes vary up to 100 feet in width; and while many are continuously profitable for long distances, the general characteristic of ore-occurrence is in lenticular masses or lenses along a general line.

The character of the ore is indicated by the following determinations of free silica from average samples of oxidized ore:

Mine.	Silica. Per cent.
Hannan's Brownhill,	18.21
Lake View Consols,	28.31
Boulder Main Reef,	31.06
Sons of Gwalia,	40.01

There are no other gangue-minerals of consequence, the remainder being almost all kaolin, showing that the oxidized ore is but slightly different in major composition from the country-rock. In the unoxidized ores more quartz is present; and it becomes evident that some replacement of the country-rock by quartz and the metalliferous minerals has taken place; but, as a whole, the deposits are not structurally complex, although they are so, chemically. The values often diminish gradually from a central core or source of impregnation into the surrounding rocks, the limit of the "ore" being regulated by the working costs.

Mineralization.—Below the zone of oxidization there is a wide range of accessory minerals: sulphides of iron, lead, zinc, mercury, arsenic and antimony; tellurides of gold, silver, mercury and bismuth, with many rare minerals, among them natural amalgam. The gold occurs in minor proportion as free gold, but in major value in the tellurides and sulphides, which are, as a rule, very finely disseminated through the ore, although the tellurides do occur occasionally in large patches or stringers.

Superficial Decomposition.

General Features of Alteration.—The close relation of the ores to the country-rock in chemical composition is well shown by the similarity of the oxidized ore to the altered country-rock. All lines of demarcation and the slender distinctive features of the unoxidized ores are obliterated with oxidization; and the miner has difficulty in selecting ore from waste. The oxidation and hydration of the accessory minerals, with their subsequent leaching, have produced no unusual features. The iron sulphides are the last to yield to alteration, and are usually met with above the tellurides. In the oxidized ores the iron appears as stains and stringers of hematite, and it has been said that tellurium has been detected as telluride of iron, and also that natural amalgam has been found in the oxidized ores; but it seems probable that the latter, together with the other metals, seldom withstood the invasion of oxidizing agencies.

Depth of Alteration.—The depth of alteration is exceedingly variable, not only in neighboring mines and districts, but also along the same lode. At Kalgoorlie, the maximum depth is represented by Hannan's Brownhill, where the ores are still

oxidized at 400 feet; and the minimum depth by the Kalgurli and Associated mines, where the oxidation does not extend to 50 feet. Along any one lode the limit varies greatly. Thus, in the Ivanhoe, the depression in the oxidation line is as much as 100 feet below its highest points. These highest points are occasionally above water-level; moreover, the unoxidized ores occur in isolated patches far above the lowest oxidation.

The depth of alteration has been asserted to bear no relation to water-level, or its determining cause, topography.* The Hannan's Brownhill and Associated are both upon hills, and represent opposite extremes. In the former, oxidation extends 150 feet below water-level; in the latter, it does not reach that level by 100 feet or more.

The first phenomenon, that of oxidation for short distances below water-level, is present in most deposits in other countries, but is probably more pronounced in some instances because of the lack of complete saturation below water-level. The climate here being very arid, water-level in many mines is merely indicated by dampness or a few hundred gallons of water per diem. Other causes of this phenomenon may be the possible agency of alkaline waters, or the variability of water-level from year to year, the country being subject to protracted droughts.

The second occurrence—that of unoxidized ores far above water-level—is the more unusual, and seems to find partial explanation in the character of the lodes themselves. The isolated patches may be due to impermeability of the rock; and an analogous cause may be the occurrence of the ores in lenticular masses. That is to say, the maximum width of these lenses probably represents exaggerations in the zone of fracture, and their edges a confinement of the fracture, and consequent impregnation, to a narrow line. Where only the edges of these lenses reach to the surface, as in the Kalgurli, there was offered but little channel for percolation of oxidizing agents from above, and consequently but little opportunity for alteration. Again, as in the case of the Brownhill, Lake View, and others, the wider outcrops of the lenses offered every facility for the entrance of such agents.

In comparison with the country-rock, as a rule, the oxidation

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* H. P. Wooodward, *Trans. Inst. Min. and Met.*

extends to a greater depth in the lodes, which naturally offered more favorable channels to percolation.

Effect on the Gold.—The gold in the oxidized ore is of very unusual character. It occurs in three forms: as amorphous brown powder; as "paint-gold," coating the cleavage-planes with microscopic particles; and, in rarer instances, as coarse gold. No better idea can be conveyed of the exceeding minuteness of the particles in the first two instances than the fact that the slimes, which constitute an average of over 50 per cent. of Kalgoorlie ores, are almost universally richer than the sands.

The gold in the unoxidized ores occurs in a very subordinate proportion as free gold, the major value lying in chemical combination with the tellurium and in fine dissemination through the sulphides. In the decomposition of such compounds we would expect the gold in a fine state of division. In the precipitation of gold from solutions, a product similar to the amorphous gold at Kalgoorlie is secured, suggesting the origin of the latter from the decomposition of tellurides. This minute division of the gold, together with the soft permeable nature of the kaolinized lode-stuff and country-rock, offers every opportunity for secondary action on the gold and its re-deposition, both mechanical and chemical. Upon this point hinge the peculiarities of Kalgoorlie superficial alteration, and to call attention to it is the main purpose of the present paper.

Secondary Deposition of the Gold.—This secondary deposition is in evidence in two main directions: the enrichment of the lodes from the surface and the lateral impregnation of the country-rock during decomposition. These processes would tend to act in opposite ways, one to enrich and the other to impoverish the lode. Although undoubtedly the most important of the agencies which cause these phenomena is purely mechanical, chemical reaction may also be a factor.

The climate of Western Australia being very arid, the agencies of surface-erosion are slow. The almost universal absence of ordinary marks of water-erosion, the alternating drifted sand and plains of denuded rocks, bear testimony to the greater importance of the wind. Such an agency implies the fine disintegration of particles and the consequent liberation of a greater proportion of contained gold, near the point of decomposition, than the more violent agency of water. In fact, a great proportion of the minute gold particles from the surface-

erosion of the lodes have settled near their original home, and have found their way by percolation through minute fissures and seams in the dry, permeable kaolinized rock to considerable depths, over the whole of the mining area. In sampling long cross-cuts through the country-rock in the Bank of England, Oroya, Hannan's Brownhill and Crœsus mines, few assays were secured above the 100-foot level which did not yield from a trace to 8 dwts. of gold per ton. Yet in sampling cross-cuts at greater depths, traces of gold became smaller and smaller, until, below the zone of kaolinization, any trace at all of gold in the country-rock is rare. In the Crœsus South United, a cross-cut on the 100-foot level averaged 6 dwts. per ton, no assay yielding over 10 dwts.; yet a cross-cut immediately beneath, on the 200-foot level, averaged but 1 dwt. 11 grs.

It is probable that, inasmuch as the lodes offer more accessible channels to percolation—as is evidenced by their decomposition to a greater depth than the country-rock—that they have received a much greater proportion of secondary gold. The general tendency of the ores towards enrichment at the surface is broadly proved by the universal decrease of from 50 to 75 per cent. in the value per ton of the output from all the leading mines. It must not be understood that the writer believes the mines will diminish in gold-value with increasing depth until they become barren; for these remarks apply to the oxidized ores only, and those mines which have been proved valuable below this zone have nothing to fear. Yet within the oxidized zone, at least ten mines at Kalgoorlie could be named which have decreased so much in value as to become worthless. Much has been said of the enrichment of the ores at water-level,* and the Kalgurli and Associated are universally cited in proof; but the explanation of this occurrence has already been given.

The second phenomenon, that of the lateral impregnation of the gold in the decomposed rock, is evidenced clearly by the almost general rule that the stopes are wider (often twice as wide) in the oxidized than in the unoxidized ore.

Another feature that may be noted is that the unoxidized

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* This subject has been gone into by Mr. T. A. Rickard (*Trans. Inst. Min. and Met.*, March 23, 1898) since this was written. If the author has overlooked other contributions to West Australian economic geology, his excuse must be the isolated situation of a practitioner in the "back blocks" of this region, cut off from libraries and bookstores.

ores are more easily distinguishable from the country-rock, admit of more careful selection in mining, and may maintain as large an average value per ton as the oxidized ores, yet not so many tons per fathom of stope.

The problems of the possible chemical solution of the gold during or after oxidization and its redeposition from such solution are exceedingly interesting, but the study of them involves the reconstruction of conditions wholly unknown. The alkalinity of most of the subterranean waters of the gold-fields, their contained chlorides, the organic acids of surface-waters, the presence of decomposing manganese and iron sulphides and tellurides, all, under varying conditions and combinations, could help to dissolve and precipitate gold. All these chemical reactions are called upon to prove enrichment of the ores in depth by leaching and redeposition at water-level; but, aside from the non-existence of such a general fact, the constant erosion of the surface and raising of the horizon of deposition would in the end produce an enrichment of the whole oxidized zone. Such a process may thus have aided in superficial and lateral enrichment of the lodes. The subject, at least, offers a good field for investigation and is a favorite theme for speculation. The generally-cited absence of alluvial gold from the neighborhood of the richest mines at Kalgoorlie finds ample explanation in the minute division of the particles, and does not, as claimed, evidence the recent origin of the deposits.

Normal Fissure-Veins.

The mines outside of Kalgoorlie, with a few exceptions, are of this order. The subsidiary minerals are fewer and more common than at Kalgoorlie, consisting of sulphides of iron, lead, zinc, arsenic and copper. Tellurium has been rarely found in quartz-veins. The gold in these veins is altogether different from that of Kalgoorlie, being usually coarse, and occurring often in wonderfully rich patches or pockets.

Superficial Decomposition.—The superficial decomposition offers but one general difference from that of other regions, which is in the wide-spread superficial enrichment. There are many quartz-mines of value, the exploitation of which has proved the values to continue in depth; but the bare fact that, of 161 mines "promoted" on quartz-veins in these districts, only seven or eight have proved successful, is evidence on this point; for

one must believe that the surface-indications, in the majority of these cases, gave some basis of hope. It is a delicate matter to name instances, but many occur to the author in which the mines above the 50-foot level did offer extraordinary prospects. Other proofs of superficial enrichment lie in the common occurrence, near the surface, of gold in the cleavage-planes, and loose gold along the walls.

The agencies at work to produce such an enrichment in the fissure-veins are more obscure than in the impregnations, where the minute division of the gold suggests a more comprehensive explanation; but it may be remembered that the surface of Australia is one of the oldest surfaces on the globe, and that the agencies of wind, water-percolation and perhaps chemical solution, although very slow, may have wrought, in such vast time, great results.

Biographical Notice of Theodor Richter.

BY R. W. RAYMOND, NEW YORK CITY.

(Buffalo Meeting, October, 1898.)*

GEHEIMER BERGRATH PROF. DR. HIERONYMUS THEODOR RICHTER, ex-Director of the Freiberg Mining Academy and Honorary Member of this Institute (to give him the full title, which nobody ever thought of using in his genial and unceremonious presence, and the whole of which I have omitted at the head of this notice, as adding nothing to the virtue and value of the man I knew), died on Sunday, September 25, 1898. He was born in 1825 at Dresden, and in 1843, after some practical training in pharmacy, entered the Freiberg Mining Academy, where he devoted himself chiefly to metallurgy and metallurgical chemistry, under the guidance of the celebrated Plattner, whose masterly instruction in those arts, and also, especially, in assaying and the use of the blow-pipe, has been classic authority to the present day. In 1853, Richter was ap-

* Prof. Richter's death occurred September 25, 1898; and the tidings reached me too late to permit the preparation of a suitable biographical notice before the October meeting of the Institute. The present notice is offered to take the place, in the *Transactions*, of the few remarks made by me at the first session of that meeting.

pointed chemist in the royal Saxon smelting-works, and in 1857, chief assessor of the works in the Freiberg district, in which capacity he rendered spirited and useful assistance to the wise and brilliant administration of Oberberghauptmann von Beust, under whom the Freiberg mining and metallurgical industry received a new lease of life, and reorganized itself on progressive lines. The new processes introduced at that time, largely with Richter's co-operation, acquired afterwards a world-wide importance, and carried the name and fame of Freiberg into all civilized countries.

In 1856 he began, as substitute or assistant, to lecture at the Academy on blow-pipe assaying, and to conduct the laboratory-practice in this department, of which he was appointed, in 1863, full Professor.

In 1873, upon the retirement of Prof. Fritzsche, he became professor also in general assaying and metallurgy. At about this time, also, he began to discharge many of the executive duties of the management of the Academy, of which the death or retirement of many of his older colleagues had left him the leading spirit; and in 1875 he was formally appointed Director. He performed with devotion and efficiency the functions of that office until 1896, when he retired from active work. In my judgment the work of the famous Academy was as thorough and progressive under Richter as it had been before, though it was no longer so conspicuous, partly because of the inevitable decline, with the fall in the market-value of silver, of the Freiberg mines; partly because of the establishment in other places of good schools of mining and metallurgy, in view of which the advantages offered at Freiberg were no longer in any respect unique. Indeed, when the history of that illustrious school shall be finally written, I think it will show that the institution retained much longer, and surrendered much more slowly, its intellectual and professional supremacy, because of the genius of Richter and his associates, who administered it while the exclusive advantages of its industrial importance, exceptional character and splendid traditions were irrevocably fading away.

Richter was an admirable teacher—one of those who think more of personal influence than of wholesale instruction by lectures; who fasten themselves upon the individual student, and impart to him guidance and inspiration at once—as did

his master, Plattner. Yet his literary activity and his zeal and industry in research were not dampened by the routine of instruction or administration. In 1860 and 1863 he issued the two volumes of Plattner's *Lectures on Metallurgy*, and in 1865 the fourth edition of Plattner's *Blowpipe-Assaying*—works which have not yet lost their importance (Prof. Kolbeck has edited very recently a new edition of the latter), and in which the value of Richter's editing is not destroyed by the fact that, sinking his own reputation in that of his predecessor, he modestly withheld any claim to special recognition.

An instance of his enthusiastic and unselfish enthusiasm for scientific investigation I can furnish from my own recollection of him. I was a student at Freiberg in 1859, 1860 and 1861, and it was during that period that Bunsen and Kirchhoff made known the startling revelations and results of their experiments in spectrum-analysis, of which, I think, I had the honor, as a correspondent of the *N. Y. Times*, to send one of the first, perhaps the first, of popular descriptions to this country. The distinguished discoverers of the new instrument of investigation announced at once the discovery of two new metals, caesium and rubidium, and Crookes followed with the discovery of thallium. Richter, reading of the new discovery, could not wait to procure a regularly-manufactured spectroscope, but manufactured an apparatus for himself, out of an ordinary monocular spy-glass, and a prism which he made by sticking together pieces of thin glass, so as to get the proper form, and filling the vessel thus created with sulphide of carbon, the high refractive power of which is well known. With this improvised apparatus he obtained a spectrum of considerable length and distinctness. I remember well the day (though I cannot fix the date) when he called me to his study, showed me his home-made spectroscope, and told me that, by means of it, he had discovered a new metal already. This was indium, the announcement of the discovery of which (according to the authorities now at my command) was made by Richter and his colleague, Prof. Reich, in 1863. But I left Freiberg in 1861 to return to my country and enter the Union army; and in 1863 my thoughts were engrossed by subjects quite apart from spectrum-analysis. If, therefore, the date given by the text-books be correct, Richter must have told me of his discovery long before he felt that he could safely make it known. As I do not

remember that, on the occasion to which I refer, he mentioned Prof. Reich as associated with himself in the investigation, I am led to believe that, before finally publishing the discovery which he had confided to me, he verified it with the aid of Prof. Reich. However that may be (and it is a question to be settled by additional evidence), I do not see how I can be mistaken in my recollection of the conversation to which I have referred, the date of which could not have been later than the early part of 1861. And, moreover, I do not see how I can be mistaken in another recollection of the same interview, namely, Richter's statement to me that his discovery of a new element (in certain zinc-blendes, I believe) had given enormous satisfaction to dear old Breithaupt, then the professor of mineralogy at Freiberg, and author of a text-book on mineralogy, which (if my recollection is correct) was never completely published. Breithaupt's dream was to systematize mineralogy as Linnæus had systematized botany, in genera, species and varieties with Latin names. His elaborate system was not adopted by mineralogists; but one feature of it has, perhaps, approved itself as based on a scientific foundation, namely, the assumption that a peculiar general habitus, or even (if I, as a listener to his lectures thirty-nine years ago, remember correctly) the presence of a peculiar and characteristic crystal-face, subordinate to the general form, was indicative of the presence of a special element in the chemical composition. This proposition was subsequently demonstrated for calcite by Hermann Credner, who showed experimentally that the presence of minute quantities of strontia, etc., would affect, within the limits of the hexagonal system, the peculiar habitus of crystals of calcium carbonate, artificially precipitated. Its general application may be still doubtful; but in the instance to which I now refer, I remember clearly Richter's telling me that old Breithaupt was immensely pleased, because the discovery of a new element had been made in a zinc-blende concerning which he had already declared, on the strength of crystallographic evidence, that it must be a variety (no doubt with a double Latin name) characterized by a special composition.

I met Richter again, in 1873, when I made a flying visit to my Alma Mater at Freiberg, and was received by him and others of the faculty with an enthusiastic fraternal cordiality never to be forgotten, and expressing, in undiminished meas-

ure, that life-long affectionate relation between teachers and pupils which has been for many, many years the strength and the glory of the Academy. His death severs, so far as I can now recall, the last tie of personal recollection which binds me to the dear old school—with one exception; that of Prof. Albin Weisbach (known in my time as "young" Weisbach, but doubtless older now), of whom may it be long ere I have to write an obituary notice!

The Silicon-Control of Carbon in Cast-Iron.

BY F. E. BACHMAN, BUFFALO, N. Y.

A Discussion of the Paper of Mr. Summers on "Modern Cupola-Practice," etc. (See page 396.)

(Buffalo Meeting, October, 1898.)

ALTHOUGH it has been apparent to me for a long time that too great weight was currently given to the silicon-contents of foundry-iron, and that the theory of the control by silicon of the carbon-contents of cast-iron is unsound, or that, at best, the influence of silicon is too small to be of much practical value as a guide in foundry-work, it never seemed to me desirable to enter the lists against either the "silicon-fad" or the theory, for the reason that I had neither the time nor the facilities to prove my position, nor any better theory to offer. But since Mr. Summers has taken the step, it seems but right that I should give such facts as are at hand in support of his contention. At the same time I do not think these are enough to prove the case if the contrary proposition were not itself, as it seems to me, based on insufficient evidence.

Both Keep's and Turner's conclusions seem to be drawn from irons carrying less than the normal amount of carbon; Turner's "base" being, moreover, a steel rather than a pig-iron, and both the white base and the ferro-silicon of Keep being so low in carbon that compounds of them would not give a material approaching the carbon-contents of a normal casting, and having, therefore, little value as showing the effects of silicon on foundry-irons, even if his samples were treated under the same conditions as those to which pig-metal is subjected in making castings.

If Mr. Keep's compounds were formed by melting in graphite or other crucibles, and the resulting composition (as to elements other than silicon) was estimated, or even (as seems to be the case) neglected, the carbon-contents and relations are open to doubt, and all conclusions thus reached are questionable.

I submit herewith a series of 138 analyses of the pig-iron of the Buffalo furnace, made in the same furnace during a period of three years, out of ore-mixtures varying slightly from year to year, as experience showed to be desirable to improve the quality of the product, together with a few analyses of other standard irons from different sections of the country, which are added for the purpose of showing the difference in composition of the several irons, and, at the same time, proving that, although the Buffalo iron seems to carry more carbon, and a greater proportion of graphite to total carbon, than the average pig-metal, the difference is not so great as to be abnormal, and to prevent silicon from having the same effect in this as in other irons.

The Buffalo furnace analyses are all of foundry-grades. We try to make nothing else; and we have paid so little attention to the composition of the lower grades that not enough analyses of the latter are available to permit intelligent discussion.

In making up the tables, consecutive analyses have been taken, whether of good or of "off" iron, both of pig-irons and of castings.

It will be noticed that, in these tables, the Buffalo iron is divided according to grade and percentage of silicon; the averages from the first set of tables (I. to XXI.) are given in Tables XLIII. to L.; and in Table LI. these averages are arranged, for greater convenience, according to silicon.

I think it is acknowledged that, given a normal furnace, with the same burden, the same lime-charge and the same heat of blast, the silicon-content of the pig is the gauge of the furnace-heat, and that the causes which produce silicon produce graphite, to the point of saturation; therefore, we should expect higher graphite with higher silicon.

The irons in these tables carrying less than 1.50 per cent. of silicon were all "special" irons, as will be seen from their lower phosphorus-contents. They are therefore not strictly comparable with the others. But, setting these aside, I doubt if there is any tool sensitive enough to tell the difference in cast-iron

having the carbon here given with the highest and lowest silicon-averages; and I am very certain that the 3.14 per cent. silicon iron will dull a tool much faster than the 0.75 per cent. However, as the average casting does not contain less than 1.50 per cent. of silicon, only the irons here given as containing 1.50 per cent., and upward, of silicon are comparable in practical work; and no one who has analyzed, and also tooled, even two castings will venture to say that the higher-silicon iron is more desirable for machinery-castings, whatever may be its value for very small work, where fluidity is of great importance.

Turning to Table LII., composed of the averages of 97 castings, which were good, bad and indifferent, made for all purposes, we find the manganese to vary between 0.37 and 0.50 per cent.—a range which is generally acknowledged to be too small to affect the combined carbon. Sulphur and silicon, therefore, are practically the only elements affecting the carbon-contents.

As the silicon increases from 1.50 to 3 per cent., we find a gradual decrease of the combined and total carbon, amounting to 0.158 per cent. of combined and 0.207 per cent. of total carbon. The ratio of graphitic to total carbon, as shown in the line headed "Percentage of graphitic carbon," does not follow the silicon-contents at all, although the 2.86 per cent. silicon-average shows 3.91 per cent. more of the total carbon in the form of graphite than does the 1.62 per cent. silicon-average. On the other hand, taking the whole table, 3.32 per cent. silicon-average contains but 1.10 greater percentage of total carbon in graphitic form than the 1.62 per cent. silicon-average, a difference too small to confirm the theory that it is due to the silicon. Moreover, the 4.10 per cent. silicon-average actually shows a smaller proportion of graphitic carbon than the 1.62 per cent. silicon-average.

According to these series of analyses, if silicon has any influence on the carbon-contents of either castings or pig-iron, that influence is too small to be of any practical value above a silicon-percentage of from 1.50 to 1.75.

It is very questionable whether the smaller amount of fuel used, and the lower temperature required, to produce irons containing less than 1.50 to 1.75 per cent. of silicon, has not more influence on the proportion of graphitic carbon than the

silicon-contents of the iron. About 250 pounds more coke is required per ton to make a 2 per cent. than a 1 per cent. silicon-iron, and 350 pounds more to make a 3 per cent. than a 2 per cent. silicon-iron, the furnace using the same ore-mixture, operating with a slag of the same composition and with same temperature of blast.

Looking only at practical results in ordinary foundry-work, for which we can compare only such silicon-percentages as are found in every-day practice, namely, 1.50 to 3 per cent., we find

Si, ...	1.62	1.90	2.12	2.37	2.61	2.86
Graphitic carbon, .	2.959	2.935	2.9125	2.9366	2.856	2.91

This almost shows a gradual decrease of graphitic carbon, and the lowest graphitic carbon with the highest silicon.

Upon inspection of the analyses of the castings, there seems to be, between the sulphur-contents and carbon-relations, a close relation, which shows more in individual cases than in the averages; while, in the pig-iron analyses, we find that the opener grades, No. 1 and No. 2x., carry the greater amounts of total carbon, graphite and sulphur.

I have often noticed that it is much easier to make No. 1 pig-iron with 0.03 to 0.04 per cent. sulphur than with less—which seems to be an anomaly.

The only other points suggested by these records of analyses which seem worthy of special attention are the high combined-carbon and low-silicon contents of the so-called soft irons, especially in the list of Tennessee irons, given in Table XXIX., and the high combined-carbon of the high-phosphorus irons of Table XXVIII. Of these irons, No. 3 and No. 6 are Alabama all-ore irons; the remainder are all, or largely, cinder-irons. It seems reasonable to suppose that the lower graphite-contents and higher combined-carbon in the cinder-iron may be attributed to the fact that cinder absorbs little or no carbon in the higher parts of the furnace, and that, therefore, the pieces not being disintegrated, reduction is not complete before they reach the metal bath of the hearth, still carrying a portion of their original oxygen, which reacts on the carbon of the molten iron. If this hypothesis of reaction is correct, it may explain, and justify to some extent, the once popular prejudice against cinder-mixed irons.

The sampling of the Buffalo iron, of which analyses are here

given, was done by breaking the pig in the middle and drilling in the center of the fracture. That this does not give an absolutely reliable average sample, I acknowledge; but results obtained under uniform conditions are comparable.

The following analyses from three points in the same pig show to what extent the carbon segregates:

	P.	Mn.	S.	Si.	Graph. C.	Comb. C.	Total.	Proportion of Graph C.
Sow end, .	0.47	0.38	0.016	3.12	4.18	0.16	4.34	96.36
Middle end, .	0.47	0.38	0.018	3.12	3.56	0.14	3.70	96.22
Tail end, .	0.47	0.38	0.014	3.10	3.79	0.19	3.96	95.02

This shows almost a greater variation in carbon than obtains between 1.80 and 3.14 per cent. silicon-pig.

The analyses here given were made by Mr. O. O. Laudig and Mr. Frank Hersh, as part of the regular routine-work of the laboratory; the silicon (except in ferro-silicon, etc.) being determined by solution in nitric and sulphuric acid, without purifying the silicon obtained; graphite by solution in dilute nitric acid; total carbon by solution in double chloride of copper and potassium, the carbon being burned in a current of oxygen. Mr. Laudig, following Blair, absorbed the CO_2 in caustic potash, while Mr. Hersh absorbed in barium hydrate (using a special absorption apparatus), filtered off the barium carbonate formed, washed, ignited, weighed and calculated the carbon from the result. The latter method seems to be a great improvement on absorption in caustic potash, as but 6 per cent. of the final weight is carbon, and there is no liability of error due to variation in weight of the absorption apparatus. Where the saving of time is an object, and commercial laboratory-conditions are to be satisfied, these considerations have special importance.

In the following tables, besides the ordinary chemical symbols, "G. C." is employed to indicate the percentage of graphitic carbon, "C. C." that of combined carbon, "T. C." that of total carbon (the sum of G. C. and C. C.), and "Per cent. G. C." the percentage of the T. C. which is represented by G. C.

The grades of foundry-iron mentioned in the tables are familiar in the trade here; but for the sake of foreign readers, it may be well to observe that No. 2p. is plain No. 2; No. 2s. is soft No. 2; and O. Scotch and T. Scotch are brands of Niagara iron, the initials standing for "Ontario" and "Tonawanda" respectively.

Table I.

Brand, Buffalo.
Grade, No. 1x.
(Silicon, 1 to 1.25 Per cent.)

No.	1.	2.	3.	4.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P.	0.21	0.213	0.141	0.23	0.198
Mn.	0.79	0.38	0.43	0.77	0.59
S.	0.013	0.053	0.035	0.028	0.032
Si.	1.13	1.15	1.17	1.18	1.15
G. C.	3.32	3.27	3.29	3.26	3.285
C. C.	0.68	0.21	0.49	0.58	0.490
T. C.	4.00	3.48	3.78	3.84	3.775
Per cent. G. C.	83.00	93.96	87.56	84.63	87.285

Table II.

Brand, Buffalo.
Grade, No. 1x.
(Silicon, 2 to 2.25 Per cent.)

No.	1.	2.	Average.
	Per cent.	Per cent.	Per cent.
P.	0.39	0.43	0.41
Mn.	0.70	0.43	0.56
S.	0.013	0.03	0.021
Si.	2.06	2.20	2.13
G. C.	3.20	3.43	3.315
C. C.	0.16	0.19	0.175
T. C.	3.36	3.62	3.49
Per cent. G. C.	95.23	94.76	94.986

Table III.

Brand, Buffalo.
Grade, No. 1x.
(Silicon, 2.25 to 2.50 Per cent.)

No.	1.	2.	3.	4.	5.	6.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P.	0.38	0.40	0.42	0.43	0.44	0.39	0.41
Mn.	0.72	......	0.50	0.41	0.52	0.41	0.51
S.	0.024	0.012	0.029	0.041	0.024	0.040	0.028
Si.	2.28	2.28	2.30	2.36	2.42	2.42	2.34
G. C.	3.49	3.71	3.52	3.53	3.70	3.77	3.62
C. C.	0.23	0.15	0.17	0.19	0.16	0.16	0.1766
T. C.	3.72	3.86	3.69	3.72	3.86	3.93	3.7966
Per cent. G. C.	93.92	96.08	96.47	95.14	95.85	95.93	95.348

TABLE IV.

Brand, Buffalo.

Grade, No. 1x.

(Silicon, 2.50 to 2.75 Per cent.)

No.	1.	2.	3.	4.	5.	6.	7.	8.	9.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P	0.29	0.40	0.42	0.55	0.40	0.42	0.46	0.39	0.46	0.41
Mn	0.80	0.48	0.54	0.46	0.58	0.56	0.38	0.40	0.44	0.51
S	0.031	0.024	0.044	0.032	0.036	0.025	0.029	0.047	0.034	0.034
Si	2.57	2.59	2.60	2.60	2.64	2.64	2 66	2.70	2.72	2.63
G. C.	3.52	3.25	3.68	3.75	3.38	3.84	3.29	3.69	3.90	3.588
C. C.	0.18	0.19	0.15	0.14	0.15	0.16	0.16	0.12	0.14	0.153
T. C.	3.70	3.44	3.83	3.89	3.53	4.00	3.45	3.81	4.04	3.741
Per cent. G. C.	95.14	94.47	96.08	96.40	95.75	96.00	94.81	96.85	96.53	95.910

TABLE V.

Brand, Buffalo.

Grade, No. 1x.

(Silicon, 2.75 to 3 Per cent.)

No.	1.	2.	3.	4.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P	0.44	0.30	0.44	0.40	0.39
Mn	0.56	0.81	0.53	0 60	0.62
S	0.028	0.034	0.025	0.024	0.028
Si	2.80	2.82	2.92	2.95	2.87
G. C.	3.61	2.88	3.96	3.30	3.4375
C. C.	0.15	0.20	0.14	0.14	0.1575
T. C.	3.76	3.08	4.10	3.44	3.595
Per cent. G. C.	96.01	91.56	96.58	95.93	95.619

TABLE VI.

Brand, Buffalo.

Grade, No. 1x.

(Silicon, 3 to 3.40 Per cent.)

No.	1.	2.	3.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.
P	0.46	0.39	0.40	0.42
Mn	0.50	0.42	0.88	0.60
S	0.03	0.037	0.014	0.027
Si	3.00	3.06	3.40	3.15
G. C.	3.65	3.67	3.70	3.673
C. C.	0.18	0.14	0.15	0.156
T. C.	3.83	3.81	3.85	3.829
Per cent. G. C.	95.30	96.32	96.10	95.925

TABLE VII.
Brand, Buffalo.
Grade, No. 2x.
(Silicon, 0.65 to 0.90 Per cent.)

No.	1.	2.	3.	Average.
	Per cent.	Per cent.	Per cent.	Average.
P	0.22	0.21	0.23	0.22
Mn	0.67	0.74	0.74	0.72
S	0.021	0.020	0.037	0.026
Si	0.68	0.87	0.88	0.81
G. C	3.59	2.96	3.23	3.26
C. C	0.39	0.71	0.38	0.493
T. C	3.98	3.67	3.61	3.753
Per cent. G. C.	90.20	80.65	89.47	86.864

TABLE VII. A.
Brand, Buffalo.
Grade, No. 2x.
(Silicon, 1.25 to 1.50 Per cent.)

No.	1.	2.	3.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.
P	0.20	0.21	0.19	0.20
Mn	0.83	0.98	0.39	0.73
S	0.024	0.024	0.02	0.022
Si	1.25	1.35	1.40	1.33
G. C	3.20	3.80	3.44	3.48
C. C	0.32	0.32	0.39	0.343
T. C	3.52	4.12	3.83	3.823
Per cent. G. C.	90.90	92.23	92.22	91.028

TABLE VIII.

Brand, Buffalo.

Grade, No. 2x.

(Silicon, 1.75 to 2 Per cent.)

No.	1.	2.	3.	4.	5.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent	Per cent.
P	0.34	0.31	0.31	0.31	0.38	0.33
Mn	0.59	0.89	0.89	0.90	0.56	0.76
S	0.022	0.028	0.028	0.029	0.018	0.025
Si	1.84	1.85	1.85	1.86	1.94	1.86
G. C	3.21	2.92	2.75	2.96	3.36	3.04
C. C	0.24	0.21	0.29	0.20	0.24	0.236
T. C	3.45	3.13	3.04	3.16	3.60	3.276
Per cent. G. C	93.04	93.29	90.75	93.37	93.33	92.765

TABLE IX.

Brand, Buffalo.

Grade, No. 2x.

(Silicon, 2 to 2.50 Per cent.)

No.	1.	2.	3.	4.	5.	6.	7.	Av'r'ge
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent	Per cent.	Per cent.	Per cent.
P	0.52	0.42	0.51	0.28	0.34	0.39	0.30	0.39
Mn	0.45	0.46	0.51	0.78	0.83	0.41	0.73	0.59
S	0.049	0.029	0.026	0.039	0.029	0.044	0.041	0.036
Si	2.00	2.10	2.12	2.15	2.20	2.20	2.21	2.14
G. C	3.67	3.74	3.38	3.60	3.58	3.41	3.46	3.55
C. C	0.16	0.16	0.21	0.18	0.20	0.17	0.26	0.19
T. C	3.83	3.90	3.59	3.78	3.78	3.58	3.72	3.74
Per cent. G. C	95.82	95.89	94.15	95.23	94.70	95.25	93.01	94.652

TABLE X.

Brand, Buffalo. *Grade*, No. 2x.

(Silicon, 2.25 to 2.50 Per cent.)

No.	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.	12.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P.	0.45	0.38	0.37	0.45	0 29	0 29	0.34	0.46	0.38	0.38	0.44	0.40	0.39
Mn.	0.50	0.72	0.61	0.45	0.56	0.62	0.52	0.46	0.56	0.72	0.59	0.41	0.56
S.	0.020	0.025	0.015	0.025	0.024	0.031	0.013	0.024	0.028	0.018	0.031	0.024	0.023
Si.	2.28	2.30	2.32	2.34	2.34	2.35	2.40	2.42	2.44	2.47	2.48	2.49	2.39
G. C.	3.60	3.46	3.26	3.60	3.52	3.80	3.30	3.66	3.02	3.21	3.28	3.55	3.44
C. C.	0.17	0.23	0.15	0.17	0.19	0.14	0.12	0.19	0.23	0.18	0.14	0.13	0.17
T. C.	3.77	3.69	3.41	3.77	3.71	3.94	3.42	3.85	3.25	3.39	3.42	3.68	3.61
Per cent G. C.	95.49	93.76	95.60	95.49	94.87	96.44	96.49	94.57	92.92	94.69	95.90	96.46	95.288

TABLE XI.

Brand, Buffalo. *Grade*, No. 2x.

(Silicon, 2.50 to 2.75 Per cent.)

No.	1.	2.	3.	4.	5	6.	7.	8.	9.	10.	11.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P.	0.42	0.40	0.37	0.45	0.56	0.45	0.45	0.40	0.43	0.43	0.35	0.41
Mn	0.71	0.61	0.77	0.48	0.55	0 45	0.41	0.65	0.47	0.47	0.65	0.57
S.	0 022	0.042	0 036	0.037	0.011	0.026	0.026	0.026	0.034	0.034	0.041	0.030
Si.	2.50	2.54	2.54	2.60	2.63	2.64	2.64	2.64	2.68	2.68	2.74	2.62
G. C.	3.47	3.30	3.71	3.61	3.72	3.60	3 60	3.30	3.61	3.61	3.38	3.537
C. C.	0.18	0.18	0.19	0.19	0.15	0.18	0 18	0.17	0.19	0.19	0.21	0.182
T. C.	3.65	3.48	3.90	3 80	3.87	3.78	3.78	3.47	3.80	3.80	3.59	3.719
Per cent. G. C.	95.06	94.82	95.12	95.00	96 12	95 23	95.23	95.10	95.00	95.00	91.36	95.106

TABLE XII.

Brand, Buffalo.

Grade, No. 2x.

(Silicon, 2.75 to 3 Per cent.)

No.	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	Average.
	Per cent.	Per cent	Per cent	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent	Per cent.
P	0.28	0.39	0.42	0.39	0.32	0.55	0.43	0.41	0 46	0.36	0.40
Mn	0.78	0.40	0.52	0.80	0.75	0.50	0.58	0.52	0.54	0 52	0.59
S	0.049	0.032	0.035	0.015	0.01	0 034	0.039	0 037	0.030	0.015	0.035
Si	2.77	2 82	2.83	2.85	2.86	2.86	2 88	2 95	2.96	2.97	2 87
G. C	3 76	3.58	3.48	3 25	3.82	3.58	3.51	3.01	3 48	3.68	3 515
C C	0.17	0.17	0.15	0.15	0.14	0.19	0.15	0.18	0.19	0 17	0.166
T. C	3.93	3.75	3.63	3.40	3.96	3.77	3.66	3.19	3.67	3 85	3.681
Per cent. G. C	95.67	95.47	95.86	95.59	96.46	94.96	95.90	94.39	94.31	95.59	95.219

TABLE XIII.

Brand, Buffalo.

Grade, No. 2x.

(Silicon, 3 to 3.30 Per cent.)

No.	1.	2.	3.	4.	5.	6.	7.	Aver'ge
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent	Per cent.
P	0.40	0.43	0.39	0.46	0.30	0.39	0.40	0.39
Mn	0.76	0.58	0.43	0.48	0.80	0.71	0.84	0.66
S	0 015	0.02	0 032	0.027	0.018	0.010	0 022	0.02
Si	3.04	3.14	3.15	3.16	3.18	3.27	3.30	3.17
G. C	3.86	3.80	3.56	3.58	3.45	3.62	3.60	3.638
C. C	0.19	0 19	0.14	0 18	0.28	0.14	0.19	0.187
T. C	4.05	3.99	3.70	3.76	3.73	3.76	3.79	3 825
Per cent. G. C	95.31	95.24	96.22	95.21	92.94	96.27	95.00	95.111

TABLE XIV.

Brand, Buffalo.

Grade, No. 2p.

(Silicon, .50 to 1 Per cent.)

No.	1.	2.	Average.
	Per cent.	Per cent.	Per cent.
P	0.22	0.19	0.20
Mn	0.67	0.36	0.51
S	0.030	0.041	0.035
Si	0.56	0.85	0.70
G. C	3.60	3.35	3.475
C. C	0.40	0.42	0.410
T. C	4.00	3.77	3.885
Per cent. G. C	90.00	88.86	89.446

Table XV.

Brand, Buffalo.
Grade, No. 2p.
(Silicon, 1 to 1.50 Per cent.)

No.	1.	2.	Average.
	Per cent.	Per cent.	Per cent.
P	0.18	0.20	0.19
Mn	0.43	0.89	0.66
S	0 021	0.01	0.015
Si	1.05	1.48	1.26
G. C.	3.41	3.25	3.33
C. C.	0.27	0.22	0.245
T. C.	3.68	3.47	3.575
Per cent. G. C.	92.53	93.34	93.146

Table XVI.

Brand, Buffalo.
Grade, No. 2p.
(Silicon, 1.50 to 2 Per cent.)

No.	1.	2.	3.	4.	5.	6.	7.	Aver'ge
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P	0.41	0.38	0.36	0.31	0.51	0.40	0.45	0.41
Mn	0.53	0.53	0.50	0.89	0.49	0.57	0.52	0.57
S	0 021	0.016	0.037	0.028	0.018	0.01	0.032	0 023
Si	1.60	1.61	1.84	1.85	1.90	1.90	1.92	1.80
G. C.	3.20	3.25	3.43	2.75	3.27	3.52	3.68	3.30
C. C	0.30	0.32	0.20	0.29	0.23	0.16	0.23	0 247
T. C.	3.50	3.57	3.63	3.04	3.50	3.68	3.91	3 547
Per cent. G. C.	91.43	90.52	94.49	90.46	93.14	95.65	94.12	93.036

Table XVII.

Brand, Buffalo.
Grade, No. 2p.
(Silicon, 2 to 2.25 Per cent.)

No.	1	2.	3.	4.	5.	6.	Average
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P	0.42	0.46	0.41	0.33	0.39	0.48	0.41
Mn	0.50	0.42	0.50	0.81	0.59	0.41	0.54
S	0.022	0.025	0 023	0.017	0 016	0 010	0 018
Si	2.00	2.03	2.15	2.19	2.20	2.24	2.13
G. C	3.14	3.92	3.00	3.16	3.00	3.49	3.285
C. C.	0.19	0.16	0.18	0.27	0.22	0.34	0.226
T. C.	3.33	4.08	3.18	3.43	3.22	3.83	3.511
Per cent. G. C.	94.29	96.08	94.34	93.53	93.16	91.12	93.563

TABLE XVIII.

Brand, Buffalo. *Grade*, No. 2p.

(Silicon, 2.25 to 2.50 Per cent.)

No.	1.	2.	3.	4.	5.	6	7.	8.	9.	10.	11.	Average.
	Per cent.	Per cent.	Per cent	Per cent.	Per cent.	Per cent.	Per cent	Per cent.	Per cent.	Per cent.	Per cent,	Per cent.
P.	0.39	0.38	0.29	0.42	0.45	0.39	0.37	0.45	0.46	0.42	0 50	0.41
Mn	0.69	0.72	0.95	0.44	0.50	0.42	0.62	0.52	0.54	0.39	0.47	0.57
S	0.016	0 027	0.069	0.062	0.031	0.041	0.041	0.017	0.018	0.014	0.0034	0 0308
Si	2.26	2.29	2.30	2.30	2.34	2.35	2.36	2.37	2.40	2.43	2.45	2.35
G. C.	3.41	3.40	3.80	3.11	3.30	3.49	3.39	3.63	3.40	3 42	3.45	3.436
C. C.	0.19	0.30	0.17	0.28	0.20	0.18	0.20	0.20	0.22	0.19	0.19	0.211
T. C.	3.60	3.70	3.97	3.39	3 50	3.67	3.59	3.83	3.62	3.61	3 64	3.647
Per cent. G. C.	94.72	91.89	95.72	88.11	94.28	95.09	94.43	94.80	93.92	94.73	94.75	94.187

TABLE XIX.

Brand, Buffalo. *Grade*, No. 2p.

(Silicon, 2.50 to 2.75 Per cent.)

No.	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent	Per cent	Per cent.	Per cent.	Per cent.	Per cent.
P.	0.43	0.27	0.53	0.47	0.29	0.45	0.47	0.41	0.43	0.55	0.43
Mn.	0.48	0.83	0.50	0.40	0.69	0.54	0.44	0.40	0.40	0.45	0.51
S.	0.046	0.026	0.023	0.033	0.011	0.026	0.025	0.029	0 026	0.029	0.027
Si.	2.50	2.53	2.54	2.59	2.60	2.61	2 64	2.70	2.70	2.70	2.61
G. C.	3.36	3.06	3.52	3.51	3.35	3.39	3.59	3.31	3.56	3.56	3.421
C. C.	0.20	0.21	0.25	0.18	0.19	0.20	0.12	0.20	0.20	0.17	0.192
T. C.	3.56	3.27	3.77	3.69	3.54	3.59	3.71	3.51	3.76	3.73	3.613
Per cent. G. C	94.35	93.58	93.35	95.12	94.66	94.43	96.77	93.45	94.68	94.48	94.686

Table XX.

Brand, Buffalo.
Grade, No. 2p.
(Silicon, 2.75 to 3 Per cent.)

No.	1.	2.	3.	4.	5.	6.	7.	8.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent	Per cent	Per cent.	Per cent.
P	0.44	0.43	0.43	0.45	0.46	0.44	0.37	0.39	0.43
Mn	0.47	0.48	0.54	0.52	0.60	0.58	0.81	0.80	0.60
S	0 015	0 016	0.025	0 030	0.02	0 035	0.011	0.017	0.021
Si	2.76	2.80	2.80	2.86	2.89	2.93	2.93	2.97	2.87
G. C.	3.38	3.60	3.64	3.11	3.40	3.45	3.42	3.92	3.490
C. C.	0·22	0.18	0.20	0.20	0.20	0.19	0.14	0.18	0.1887
T. C.	3.60	3.78	3.84	3.31	3.60	3.64	3.56	4.10	3.6787
Per cent. G. C.	93.88	95.23	94.79	93.92	94.44	94.78	96.06	95.61	94.863

Table XXI.

Brand, Buffalo.
Grade, No. 2p.
(Silicon, 3 to 3.25 Per cent.)

No.	1.	2.	3.	4.	5.	6.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P	0.41	0.33	0.46	0.39	0.28	0.30	0.36
Mn	0.82	0.96	0.53	0.66	0.87	0 80	0.77
S	0 019	0.035	0.03	0.016	0.021	0.019	0.023
Si	3.00	3.03	3.10	3.14	3.18	3.21	3.11
G. C	3.68	3.22	3.68	3.82	2.62	3 45	3.411
C. C.	0.22	0.28	0.18	0.16	0.43	0.30	0.261
T. C.	3.90	3.50	3 86	3.98	3.05	3.75	3.672
Per cent G. C.	94.35	92.00	95.33	95.98	85.90	92.00	92.810

Table XXII.

Brand, Thomas.

Grade, No.	1x.	1x.	2x.	2x.	2x.	2x.	2p.	2p.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P	1.25	1.10	0.95	0.90	1.10	0.71	0.93	1.00
Mn.	0.10	0.14	0.31	0.10	0.14	0.19	0.13	0.10
S.	0.010	0.007	0 008	0.014	0 007	0 010	0.008	0.014
Si	2.08	2.67	1.94	2.34	2.67	2.73	1.64	2.14
G. C.	3.69	3.00	3.00	3.55	3.18	3.18	3.38	3.35
C. C.	0.16	0.22	0.62	0.22	0.23	0.12	0.52	0.15
T. C.	3.85	3.22	3.62	3.77	3.41	3.30	3.90	3.50
Per cent. G. C.	95.61	93.16	82.88	94.16	93.25	96.36	86.66	95.71

TABLE XXIII.

Brand, Niagara.

Grade, No.	2x.	2x.	2x.	2p.	2p.	T. Scotch.	O. Scotch.	T. Scotch.	T. Scotch.	T. Scotch.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P......................	0.35	0.27	0 35	0.50	0.29	0.87	1.16	0.43	0.54	0.65
Mn....................	0.57	0.64	0.94	0.73	0 78	0 44	0.45	0.44	0.78	0.72
S........................	0.022	0 017	0 021	0 070	0.028	0.067	0 017	0 046	0.038	0.008
Si......................	1.51	1.72	1.81	2.17	2.84	2.82	2.84	3.00	3.69	4.18
G. C..................	3 55	3.15	2.72	3.27	3.27	3 35	2.94	3.61	3.21	2.97
C. C..................	0 27	0 26	0.18	0.21	0.30	0.21	0.16	0.20	0.29	0.17
T. C..................	3.82	3.41	2.90	3.48	3.57	3.56	3.10	3.81	3.50	3.14
Per cent G. C.....	92.90	92.62	90.34	93.71	91.04	94.10	94.84	94.75	91.14	94.58

TABLE XXIV.

Valley Scotch Irons.

Brand.	Grade, No.	P.	Mn.	S.	Si.	G. C	C C.	T. C.	G. C.
		Per cent.	Per cent.	Per cent	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Ohio Scotch........	2	0 48	0.56	0.049	1.43	2.98	0.29	3.27	91.10
Cherry Valley....	1	0 86	0.74	0.016	1.94	2.83	0.36	3.19	88.71
Cherry Valley....	2	0.70	0.80	0.023	1.75	2.80	0.25	3.05	93.18
Hazleton............	1	0.71	0.48	0.032	2.06	3.20	0.20	3.40	94.11
Hazleton............	1	0.64	0.47	0.030	2.28	3.31	0.29	3.60	91.94

TABLE XXV.

High-Silicon, Silvery, Ferro-Silicons, Etc.

Brand.	Grade, No.	P.	Mn.	S.	Si	G. C.	C. C.	T C.	G. C.
		Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Ashland...	1	1.45	0.63	0.021	4.94	2.90	0.18	3.08	94.15
"	1	1.66	0.46	0.007	5.57	2.06	0.16	2.22	92.18
"	1	1.50	0.33	0.008	5 88	2.60	0.25	2.85	91.23
"	1	1.21	0.58	0.010	7.40	2.30	0.12	2.42	95.04
"	1	1.67	0.86	0.014	7.60	2.21	0.20	2.41	91.29
"	1	1.67	0.66	0 05	8.10	1.90	0.15	2.05	92.68
"	1	1.43	0.40	0.024	8.10	1.56	0.13	1.69	92.37
"	Silico-spiegel	0.105	18.90	0.010	12.10	1.21	0.49	1.70	71.17
"	Ferro-silicon.	0.093	0.41	....	12.24	1.10	0.13	1.23	89.43
"	Ferro-silicon.	0.105	0.17	0.05	12.83	1.36	0.12	1.48	91.89
Lawrence.		1.58	0.70	0.028	4.93	2.47	0.08	2.55	96.86
Star.........		0.92	0.12	0.008	7.97	1.84	0.18	2.02	91.09
Star.........		0.96	0.42	0.009	9.33	2.13	0.09	2.22	95.95
Bessie......		0.92	0.42	0.019	6.80	2.47	0.05	2.52	98 01
Globe......		1.53	0.57	0.009	5.33	2.30	0.10	2.40	95.83

Table XXVI.

Alabama Irons.

No. Grade.	1. 2s.	2. 2s.	3. ...	4. 2s.	5. 2s.	6. 1.	7. 2s.	8. 2s.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P	0.69	0.87	0.47	0.84	0.72	0.74	0.85	0.82
Mn	0.51	0·30	0.42	0.24	0.21	0.31	0.34	0.20
S	0.052	0.010	0.035	0.013	0.010	0.014	0.024	0.008
Si	2.49	2.60	2.98	3.00	3.02	3.23	3.25	3.21
G. C.	3.49	3.33	3.30	3.29	3.44	2.96	3.66	2.95
C. C.	0.27	0.27	0.15	0.14	0.20	0.17	0.19	0.38
T. C.	3.76	3.60	3.45	3.43	3.64	3.13	3.85	3.33
Per cent. G. C.	92.82	92.50	95.68	95.92	94.56	94.88	95.06	88.58

Table XXVII.

Southwest Virginia Irons.

No. Grade.	1. 2.	2. 1.	3. 2.	4. 2p.
	Per cent.	Per cent	Per cent.	Per cent.
P	0.46	0.82	0.35	0.86
Mn	1.30	0.84	0.70	0.85
S	0.024	0.013	0.016	0.020
Si	1.90	1.96	2.12	3.45
G. C	3.00	3.73	3.11	3.53
C. C	0.72	0.16	0.20	0.23
T. C	3.72	3.89	3.31	3.76
Per cent. G. C.	80.64	95.63	93.96	93.88

Table XXVIII.

High-Phosphorus Irons.

No. Grade.	1. G. F.	2. G F.	3. 2s.	4. 2p.	5. 2	6. 2s.	7. 2.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P	2.65	2.66	2.11	2.54	2.74	2.13	3.64
Mn	0.75	0.78	0.46	0.78	0.97	0.48	1.17
S	0.01	0.012	0.022	0.010	0.008	0.032	0.03
Si	0.90	1.02	1.47	1.73	1.88	2.20	2.37
G. C.	2.00	2.21	2.70	2.38	2.67	2.95	2.44
C. C.	1.26	1.02	0.28	0.72	0.47	0.18	0.49
T. C	3.26	3.23	2.98	3.10	3.14	3.13	2.93
Per cent. G.C.	61.04	65.29	90.60	76.77	85.03	94.21	83.27

TABLE XXIX.

Tennessee Irons.

No. Grade.	1. 2s.	2 1s	3 3f.	4. 2s.	5 2f.	6. 1s.	7. 2s.	8. 3f.	9. 2f.	10 1f.	11. 1s.	12. 1f.	13. 2f.	14. 2s.	15. 1s.	16. 2s.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent	Per cent.	Per cent.
P	1.18	1.19	1.19	1.10	1.19	1.20	0.77	1.18	1.10	1.10	1.12	1.07	1.12	1.17	1.15	1.16
Mn	1.10	1.16	0.35	0·53	0.35	0.50	0.16	0.65	0.55	0.30	0.41	0.35	0.47	0.40	0.40	0.56
S	0.009	0.009	0.010	0.010	0.008	0.008	0 014	0.008	0 009	0.008	0.012	0 016	0.024	0.005	0.010	0 008
Si.	1.23	1.29	1.50	1.53	1.56	1.60	1.87	1.90	1.96	2.06	2.09	2.13	2.19	2.30	2.43	2.44
G. C.	2.61	2.93	2.51	3.01	3.25	3.12	2.80	2.96	3.11	3.03	2.20	2.84	2.62	3.01	3.34	2.88
C. C.	0.65	0.78	0.29	0.65	0.16	0.46	0.70	0.28	0.28	0.20	0.59	0.48	0.29	0.53	0.39	0.60
T. C.	3.26	3.71	2.80	3.66	3.41	3.58	3.50	3.24	3.39	3.23	2.79	3.32	2.91	3.54	3.73	3.48
Per cent. G. C.	80.06	78.97	89.67	77.20	95.31	87.15	80.00	91.35	91.74	93.18	78.85	85.54	90.03	85.03	89.58	81.35

TABLE XXX.

Charcoal-Irons.

No.	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.	12.	13.	14.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P	0.20	0.21	0.20	0.19	0.22	0.20	0.25	0.26	0.25	0.24	0.19	0.24	0.75	0.095
Mn	0.65	0.87	0.87	0.44	0.86	0.76	0 53	0.61	0.40	1.66	1.02	1.07	0.49	0.80
S	0.009	0.010	0.017	0.010	0.010	0.009	0.010	0.010	0 011	0.008	0.005	0.008	0.019	0.003
Si.	1.03	1.07	1.10	1.20	1.24	1.35	1.36	1.42	1.55	1.56	1.56	1.73	1.84	2.24
G. C.	3.12	3.10	3.20	3.07	3.21	3.20	2.56	2.56	2.60	3.05	3.20	3.35	2.82	3.75
C. C.	0.64	0.71	0.70	0.62	0 58	0.62	0.56	0.52	0.52	0.55	0.48	0.95	0.33	0.30
T. C	3.76	3.81	3.90	3.69	3.79	3.82	3.12	3.08	3.12	3.60	3.68	4.30	3.15	4.05
Per cent. G. C.	84.65	81.36	82.56	81.00	84.65	83.76	82.05	83.71	83.33	84.72	86.95	77.90	89.52	92.59

Table XXXI.

Castings.

(Silicon, 0.7 to 1.25 Per cent.)

No	1.	2.	3.	4.	5.	6.	Average
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P................	0.37	0.51	0.70	0.55	0.33	0.29	0.46
Mn..............	0.53	0.46	0.46	0.35	0.47	0.66	0.49
S................	0.123	0.179	0.114	0.087	0.065	0.076	0.107
Si...............	0.71	0.88	1.07	1.08	1.13	1.15	1.00
G. C...........	2.21	2.64	2.72	2.81	2.82	3.31	2.751
C. C...........	0.40	0.92	0.56	0.46	0.50	0.60	0.563
T. C...........	2.61	3.56	3.28	3.27	3.32	3.91	3.314
Per cent. G.C.	84.67	74.15	82.92	85.93	84.94	84.91	82.767

TABLE XXXII.

Castings.

(Silicon, 1.25 to 1.50 Per cent.)

No.	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.	12.	13.	14.	Average.
	Per cent.	Per cent	Per cent	Per cent.	Per cent.	Per cent	Per cent	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent	Per cent.	Per cent.
P	0.49	0.60	0.49	0.75	0.69	0.87	0.22	0.30	0.67	0.36	0.72	0.67	0.68	0.80	0.59
Mn	0.40	0.65	0.49	0.50	0.52	0.53	0.38	0.38	0.20	0.45	0.68	0.21	0.30	0.63	0.45
S.	0.097	0.074	0.066	0.091	0.060	0.063	0.151	0.080	0.19	0.074	0.126	0.19	0.075	0.063	0.100
Si	1.25	1.29	1.29	1.30	1.34	1.36	1.40	1.43	1.45	1.45	1.45	1.46	1.48	1.49	1.38
G. C.	2.48	2.47	3.14	2.41	2.85	2.94	2.60	3.35	1.70	2.92	3.08	2.30	3.31	3.27	2.773
C. C.	0.31	0.68	0.76	0.71	0.50	0.55	0.80	0.55	0.70	0.71	0.47	0.12	0.49	0.34	0.549
T. C.	2.79	3.15	3.90	3.12	3.35	3.49	3.40	3.90	2.40	3.63	3.55	2.42	3.80	3.61	3.322
Per cent. G. C.	88.88	78.41	80.51	77.24	85.07	74.50	76.47	85.89	70.81	80.44	86.76	95.00	86.87	90.58	83.474

TABLE XXXIII.

Castings.

(Silicon, 1.50 to 1.75 Per cent.)

No.	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P	0.78	0.66	0.48	0.78	0.72	0.69	0.50	0.44	0.80	0.70	0.82	0.67
Mn	0.30	0.41	0.33	0.46	0.44	0.43	0.55	0.50	0.39	0.56	0.26	0.42
S	0.091	0.069	0.089	0.078	0.080	0.097	0.051	0.080	0.101	0.069	0.078	0.080
Si	1.57	1.57	1.57	1.60	1.60	1.60	1.62	1.65	1.67	1.68	1.73	1.62
G. C	3.07	3.08	3.00	2.90	2.84	2.82	2.90	2.92	2.89	3.08	3.05	2.959
C. C	0.51	0.48	0.46	0.43	0.78	0.29	0.59	0.50	0.64	0.58	0.28	0.5036
T. C	3.58	3.56	3.46	3.33	3.62	3.11	3.49	3.42	3.53	3.66	3.33	3.4627
Per cent. G. C.	85.75	86.51	86.70	87.08	78.45	90.67	83.09	85.67	81.87	81.91	90.90	85.456

TABLE XXXIV.

Castings.

(Silicon, 1.75 to 2 Per cent.)

No.	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.	12.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P.	0.78	0.96	0.68	0.79	0.66	0.81	0.87	0.67	0.92	0.71	1.05	0.56	0.79
Mn.	0.36	0.31	0.46	0.20	0.40	0.64	0.43	0.26	0.20	0.56	0.35	0.36	0.38
S.	0.083	0.075	0.108	0.046	0.106	0.073	0.048	0.069	0.034	0.040	0.050	0.08	0.068
Si.	1.78	1.82	1.85	1.87	1.89	1.90	1.91	1.92	1.96	1.96	1.96	1.98	1.90
G. C.	3.14	2.43	3.27	3.11	2.90	3.01	2.58	2.67	2.95	3.26	3.00	2.90	2.935
C. C.	0.47	0.84	0.31	0.41	0.37	0.37	0.84	0.60	0.45	0.56	0.32	0.32	0.488
T. C.	3.61	3.27	3.58	3.52	3.27	3.38	3.42	3.27	3.40	3.82	3.32	3.22	3.423
Per cent. G. C.	86.98	74.71	91.06	88.35	88.68	89.05	75.43	81.64	86.76	85.33	90.36	90.06	85.738

TABLE XXXV.

Castings.

(Silicon, 2 to 2.25 Per cent.)

No.	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.	12.	Average.
	Per cent.	Per cent	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P.	0.74	0.59	0.81	0.64	0.63	0.65	1.16	1.00	1.02	0.60	0.42	0.700	0.75
Mn.	0.51	0.59	0.39	0.30	0.40	0.67	0.27	0.43	0.26	0.36	0.47	0.600	0.43
S.	0.096	0.041	0.09	0.102	0.084	0.049	0.048	0.046	0.034	0.058	0.113	0.050	0.068
Si.	2.00	2.02	2.06	2.07	2.09	2.11	2.16	2.14	2.15	2.17	2.21	2.23	2.12
G. C.	2.67	3.06	2.70	2.60	3.14	3.43	2.42	2.92	3.26	2.90	3.25	2.60	2.9125
C. C.	0.36	0.36	0.55	0.63	0.38	0.36	0.40	0.42	0.32	0.43	0.27	0.75	0.4358
T. C.	3.03	3.42	3.25	3.23	3.52	3.79	2.82	3.34	3.58	3.33	3.52	3.35	3.3483
Per cent. G. C.	88.11	89.47	83.07	80.45	89.20	90.50	85.81	87.42	91.06	87.08	92.32	77.61	86.954

TABLE XXXVI.

Castings.

(Silicon, 2.25 to 2.50 Per cent.)

No.	1.	2.	3	4.	5.	6.	7.	8.	9.	10.	11.	12.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P.	0.58	0.94	0.96	0.85	0.70	0.42	0.83	1.00	0.70	0.71	0.90	0.86	0.79
Mn.	0.36	0.42	0.50	0.48	0.38	0.75	0.42	0.78	0.32	0.40	0.45	0.45	0.48
S.	0.109	0.072	0.072	0.048	0.093	0.060	0.082	0.042	0.058	0.093	0.080	0.072	0.073
Si	2.25	2.31	2.27	2.32	2.32	2.35	2.37	2.41	2.44	2.46	2.46	2.48	2.37
G. C.	2.99	2.60	2.90	2.56	3.37	2.90	2.91	2.91	2.89	3.10	3.18	2.93	2.9366
C. C.	0.22	0.63	0.50	0.86	0.28	0.30	0.51	0.24	0.44	0.58	0.46	0.42	0.4533
T. C.	3.21	3.23	3.40	3.42	3.65	3.20	3.42	3.15	3.33	3.68	3.64	3.35	3.39
Per cent. G. C.	93.11	80.49	85.29	74.85	92.49	90.62	85.08	92.37	86.78	84.21	87.36	87.46	86.648

TABLE XXXVII.

Castings.

(Silicon, 2.50 to 2.75 Per cent.)

No.	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	Average.
	Per cent.	Per cent.	Per cent	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P.	0.85	0.86	0.62	1.04	0.58	0.71	0.84	0.56	0.60	0.81	0.75
Mn.	0.45	0.43	0.35	0.44	0.38	0.46	0.50	0.40	0.42	0.30	0.41
S.	0.075	0.099	0.080	0.165	0.0[illegible]5	0.123	0.063	0.049	0.086	0.095	0.090
Si.	2.52	2.53	2.56	2.56	2.63	2.63	2.67	2.68	2.69	2.72	2.61
G. C.	2.86	2.80	2.88	2.32	3.21	2.90	2.84	3.39	2.90	2.46	2.856
C. C.	0.56	0.32	0.38	0.34	0.24	0.39	0.38	0.26	0.70	0.30	0.387
T. C.	3.42	3.12	3.26	2.66	3.45	3.29	3.22	3.65	3.60	2.76	3.243
Per cent. G. C.	83.62	89.74	88.34	87.21	93.01	88.14	88.19	92.87	80.55	89.13	88.066

Table XXXVIII.

Castings.

(Silicon, 2.75 to 3 Per cent.)

No.	1.	2.	3.	4.	5.	6.	7.	8.	9.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P	0.87	0.70	0.94	0.58	0.15	0.90	0.69	0.74	0.87	0.71
Mn	0.47	0.23	0.20	0.80	0.32	0.62	0.74	0.53	0.33	0.47
S	0.094	0 063	0.042	0.072	0.156	0.050	0.050	0 065	0.077	0.074
Si	2.75	2.76	2.76	2.85	2.85	2.86	2.95	2.96	2.97	2.86
G. C.	2.82	2.95	3.22	2.87	2.84	2.58	3.00	3.38	2.53	2.91
C. C.	0.27	0.53	0.30	0.26	0.52	0.27	0.28	0.26	0.43	0.346
T. C.	3.09	3.48	3.52	3.13	3.36	2.85	3.28	3.64	2.96	3.256
Per cent. G. C.	91.26	84.71	91.47	91.69	84.52	90.53	91.46	95.48	85.47	89.370

Table XXXIX.

Castings.

(Silicon, 3 to 3.25 Per cent.)

No.	1.	2.	3.	4.	5.	6.	7.	Average.
	Per cent	Per cent.	Per cent.	Per cent	Per cent.	Per cent.	Per cent	Per cent.
P	0.86	0.85	0.85	0.86	0.34	0.61	0.90	0.75
Mn	0.34	0.52	0.30	0.40	1.16	0.46	0.33	0.50
S	0.085	0.06	0.108	0.056	0.061	0.047	0.08	0.071
Si	3.00	3.02	3.09	3.10	3.10	3.10	3.14	3.08
G. C.	2.73	3.55	2.84	2.82	3.20	3.40	3.14	3.097
C. C.	0.50	0.25	0.49	0.44	0.30	0.26	0.20	0.3486
T. C.	3.23	3.80	3.33	3.26	3.50	3.66	3.34	3.4456
Per cent. G. C.	83.74	93.42	85.28	85.34	91.44	92.89	94.01	89.883

Table XL.

Castings.

(Silicon, 3.25 to 3.50 Per cent.)

No.	1.	2.	3.	4.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
P	0.90	0.76	0.95	0.95	0.89
Mn	0.30	0.39	0.40	0.40	0.37
S	0.099	0.142	0.102	0.104	0.112
Si	3.25	3.25	3.39	3.40	3.32
G. C.	2.80	2.40	3.05	2.96	2.802
C. C.	0.47	0.68	0.28	0.30	0.432
T. C.	3.27	3.08	3.33	3.26	3.235
Per cent. G. C.	85.62	77.92	91.29	90.79	86.549

TABLE XLI.

Castings.

(Silicon, 3.50 to 3.75 Per cent.)

No.	1.	2.	3.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.
P	0.16	0.76	0.97	0.63
Mn	0.66	0.22	0.33	0.40
S	0.066	0.070	0.066	0.067
Si	3.53	3.65	3.72	3.63
G. C	3.10	2.50	2.94	2.857
C. C.	0.31	0.31	0.24	0.287
T. C.	3.41	2.81	3.18	3.144
Per cent. G. C.	90.91	88.96	91.82	90.871

TABLE XLII.

Castings.

(Silicon, 4 to 4.25 Per cent.)

No.	1.	2.	Average.
	Per cent.	Per cent.	Per cent.
P	1.14	1.01	1.07
Mn	0.33	0.42	0.37
S	0.091	0.107	0.099
Si	4.07	4.13	4.10
G. C	2.72	2.55	2.635
C. C	0.40	0.53	0.465
T. C	3.12	3.08	3.10
Per cent. G. C.	87.17	82.79	85.00

TABLE XLIII.

Pig-Iron Averages.

(Silicon, .50 to 1 Per cent.)

Grade, No.	2x.	2p.	Average.
	Per cent.	Per cent.	Per cent.
P	0.22	0.20	0.21
Mn	0.72	0.51	0.61
S	0.026	0.035	0.03
Si	0.81	0.70	0.75
G. C	3.260	3.475	3.367
C. C	0.493	0.410	0.451
T. C	3.753	3.885	3.818
Per cent. G. C.	86.864	89.446	88.155

TABLE XLIV.

Pig-Iron Averages.

(Silicon, 1 to 1.50 Per cent.)

Grade, No.	1x.	2x.	2p.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.
P	0.20	0.20	0.19	0.20
Mn	0.59	0.73	0.66	0.66
S	0.032	0.022	0.015	0.023
Si	1.15	1.33	1.26	1.25
G. C	3.285	3.480	3.330	3.365
C. C	0.490	0.343	0.245	0.359
T. C	3.775	3.823	3.575	3.724
Per cent. G. C.	87.285	91.028	93.146	90.453

Table XLV.

Pig-Iron Averages.

(Silicon, 1.50 to 2 Per cent.)

Grade, No.	2x.	2p.	Average.
	Per cent.	Per cent.	Per cent.
P	0.33	0.41	0.37
Mn	0.76	0.57	0.66
S	0.025	0.023	0.024
Si	1.86	1.80	1.83
G. C	3.040	3.300	3.170
C. C	0.236	0.247	0.242
T. C	3.276	3.547	3.412
Per cent. G. C.	92.765	93.036	92.90

Table XLVI.

Pig-Iron Averages.

(Silicon, 2 to 2.25 Per cent.)

Grade, No.	1x.	2x.	2p.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.
P	0.41	0.39	0.41	0.40
Mn	0.56	0.59	0.54	0.56
S	0.021	0.036	0.018	0.025
Si	2.13	2.14	2.13	2.13
G. C	3.315	3.55	3.285	3.383
C. C	0.175	0.19	0.226	0.197
T. C	3.490	3.74	3.511	3.580
Per cent. G. C.	94.986	94.652	93.563	94.400

TABLE XLVII.

Pig-Iron Averages.

(Silicon, 2.25 to 2.50 Per cent.)

Grade, No.	1x.	2x.	2p.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.
P..................	0.41	0.39	0.41	0.40
Mn..............	0.51	0.56	0.57	0.55
S..................	0.028	0.023	0.031	0.027
Si................	2.34	2.39	2.35	2.36
G. C.............	3.620	3.44	3.436	3.499
C. C.............	0.1766	0.17	0.211	0.1855
T. C.............	3.7966	3.61	3.647	3.6845
Per cent. G. C.	95.348	95.288	94.187	94.941

TABLE XLVIII.

Pig-Iron Averages.

(Silicon, 2.50 to 2.75 Per cent.)

Grade, No.	1x.	2x.	2p.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.
P..................	0.41	0.41	0.43	0.42
Mn..............	0.51	0.57	0.51	0.53
S..................	0.034	0.030	0.027	0.030
Si................	2.63	2.62	2.61	2.62
G. C.............	3.588	3.537	3.421	3.515
C. C.............	0.153	0.182	0.192	0.176
T. C.............	3.741	3.719	3.613	3.691
Per cent. G. C.	95.910	95.106	94.686	95.234

TABLE XLIX.

Pig-Iron Averages.

(Silicon, 2.75 to 3. Per cent.)

Grade, No.	1x.	2x.	2p.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.
P..................	0.39	0.40	0.43	0.41
Mn...............	0.62	0.59	0.60	0.60
S..................	0.028	0.035	0.021	0.028
Si..................	2.87	2.87	2.87	2.87
G. C..............	3.4375	3.515	3.490	3.481
C. C..............	0.1575	0.166	0.189	0.171
T. C..............	3.595	3.681	3.679	3.652
Per cent. G. C.	95.619	95.219	94.863	95.234

TABLE L.

Pig-Iron Averages.

(Silicon, 3 to 3.25 Per cent.)

Grade, No.	1x.	2x.	2p.	Average.
	Per cent.	Per cent.	Per cent.	Per cent.
P..................	0.42	0.39	0.36	0.39
Mn...............	0.60	0.66	0.77	0.68
S..................	0.027	0.020	0.023	0.023
Si..................	3.15	3.17	3.11	3.14
G. C..............	3.673	3.638	3.411	3.574
C. C..............	0.156	0.187	0.261	0.201
T. C..............	3.829	3.825	3.672	3.775
Per cent. G. C.	95.925	95.111	92.810	94.615

TABLE LI.

Averages of Pig-Iron Averages.

	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Si	0.50 to 1.00	1.00 to 1.50	1.50 to 2.00	2.00 to 2.25	2.25 to 2.50	2.50 to 2.75	2.75 to 3.00	3.00 to 3.25
P	0.21	0.20	0.37	0.40	0.40	0.42	0.41	0.39
Mn	0.61	0.66	0.66	0.56	0.55	0.53	0.60	0.68
S	0.03	0.023	0.024	0.025	0.027	0.030	0.028	0.023
Si	0.75	1.25	1.83	2.13	2.36	2.62	2.87	3.14
G. C	3.367	3.365	3.170	3.383	3.499	3.515	3.481	3.574
C. C	0.451	0.359	0.242	0.197	0.1855	0.176	0.171	0.201
T. C	3.818	3.724	3.412	3.580	3.6845	3.691	3.652	3.775
Per cent. G. C	88.155	90.453	92.900	94.400	94.941	95.234	95.234	94.615
No. of samples	5	9	12	15	29	30	22	16

TABLE LII.

Averages of Castings.

	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Si	0.70 to 1.25	1.25 to 1.50	1.50 to 1.75	1.75 to 2.00	2.00 to 2.25	2.25 to 2.50	2.50 to 2.75	2.75 to 3.00	3.00 to 3.25	3.25 to 3.50	3.50 to 3.75	4.00 to 4.25
P	0.46	0.59	0.67	0.79	0.75	0.79	0.75	0.71	0.75	0.89	0.63	1.07
Mn	0.49	0.45	0.42	0.38	0.43	0.48	0.41	0.47	0.50	0.37	0.40	0.37
S	0.107	0.100	0.080	0.068	0.068	0.073	0.090	0.074	0.071	0.112	0.067	0.099
Si	1.00	1.38	1.62	1.90	2.12	2.37	2.61	2.86	3.08	3.32	3.63	4.10
G. C	2.751	2.773	2.959	2.935	2.9125	2.9366	2.856	2.91	3.097	2.802	2.857	2.635
C. C	0.563	0.549	0.5036	0.488	0.4358	0.4533	0.387	0.346	0.3486	0.432	0.287	0.465
T. C	3.314	3.322	3.4627	3.423	3.3483	3.3900	3.243	3.256	3.4456	3.235	3.144	3.100
Per cent G. C	82.767	83.474	85.456	85.738	86.954	86.648	88.065	89.370	89.883	86.549	90.871	85.000
No. of samples	6	14	11	12	12	12	11	9	7	4	3	2

DISCUSSIONS.

The Genesis of Certain Auriferous Lodes.

Continued Discussion of the Paper by Dr. John R. Don, presented at the Chicago Meeting, February, 1897. (See *Trans.*, xxvii., 564, 993.)

(Buffalo Meeting, October, 1898.)

Dr. Don (communication to the Secretary): I have to express my grateful thanks to Messrs. Le Conte, Emmons, Becker and Winslow, for their very kind remarks, presented at the Atlantic City meeting, on my paper. An investigator works practically alone and under many discouragements, at this end of the world; and nothing has encouraged me so much in my research as to find my work appreciated by men whose wide knowledge of the subject and whose eminent services to science entitle their opinions to my deepest respect.

Professor Le Conte's last sentence gives me an opportunity to express the hope that chemical analysis may in future be more generally used as an aid to observation. If any degree of finality is ever to be reached with regard to this most puzzling question, such finality will only be attained by the combination of the experimental work of the chemist with the skilled observation of the mining geologist.

Mr. Emmons, in his highly valued criticism, says:*

"Dr. Don's first and most important conclusion from his tests is that gold does not occur in the rocks of the regions investigated by him as an original constituent of the bisilicates, and that where it is found in these rocks it is associated with sulphides, mainly of iron. His inference seems to be that it cannot be original in the rock, because pyrite is necessarily a secondary constituent, that is, one introduced after the rock consolidated."

I confess that I had not in my mind as clearly as I could wish the fact that in the case of crystalline eruptive and plutonic rocks the pyrite and other sulphides may be as much a primary constituent of the rocks as the bisilicates.

This distinction between primary and secondary sulphides need not, however, be observed in the great majority of the

* *Trans.*, xxvii., 994.

country-rocks examined by me. These country-rocks may be roughly divided into four classes:

a. Those that are unmistakable sediments, more or less altered. These form the great majority of the samples analyzed; and in these cases, all sulphides must be "secondary."

b. Dike-rocks from the Upper and Lower Silurian of Victoria. None of these rocks could be obtained in the unaltered state; but those that were least altered contained no sulphides.

c. The remarks under *b* apply also to the andesites of the Thames gold-field, and the tonalite of the Charters Towers field of Queensland.

d. In the case of the gneissoid rocks of the Manipori formation of New Zealand, and the granite that probably underlies the sedimentaries of Victoria, the sulphides found in some samples may have been primary.

On page 3 of the pamphlet discussion, Mr. Emmons says:*

"It seems important to note that the lateral secretion theory which Dr. Don's tests seem to disprove is not the one that has been generally advocated in the United States; for I fancy few American geologists believe in the narrow view advocated by Sandberger, that the metals are derived necessarily from the immediately adjacent country- or wall-rock."

The introduction to Chapter V. (omitted in the condensed form of my paper) included a discussion of this point, and showed that the analysis of underlying granites and other crystalline rocks had been undertaken with a view to testing this later extension of the lateral-secretion theory. By reference to pp. 26 and 27 of my paper,† it will be seen that the crystalline rocks were chosen because they afforded an opportunity for testing, not the actual country-rock bounding the lodes, but the crystalline rock-mass that probably underlies many of the rocks in which our lodes occur.

The majority of the results noted in Table XVIII., *a*, *b* and *c*, come under this heading. Owing to the great labor involved in isolating some of the crystalline constituents of such rocks, the number of these analyses is comparatively small; but so far as these results are conclusive, they point to a source for the gold below even the crystalline rocks that may be assumed to underlie our gold-bearing sedimentaries.

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* *Trans.*, xxvii., 995.

† *Trans.*, xxvii., 589, 590.

I have to thank Mr. Emmons for his criticism* pointing out a weakness in my statement of the two alternatives on p. 34 of my paper.† The second alternative should certainly have been extended in the direction suggested by him. On page 43,‡ however, the action of the oxidizing vadose waters, in enriching vein-gold by dissolving out part of the silver, is taken into account.

Mr. Becker§ draws a clear distinction between the two kinds of pyrite associated with country-rock. This distinction seems to me to be of very great interest, and of the first importance.

Granted this distinction, and the selective osmotic action exercised by the wall-rock, a new light is thrown on many of the results obtained in my analyses. The apparently abnormal results obtained in some cases—more particularly in the Thames andesites and the country bounding the Reefton lodes—become much clearer in the light of Mr. Becker's remarks.

Mr. Becker refers also to the presence of visible particles of gold in undecomposed crystalline and eruptive rocks. Such instances are by no means rare; but is it not possible that the samples in which these occurrences are noticed may have been taken from the vicinity of lodes, so that the gold found may have been an impregnation from the lode itself? An interesting example of such an occurrence lately came under my notice. In a sample of syenitic rock taken from the vicinity of a copper-lode in Dusky Sound, on the west coast of Otago, minute specks of metallic copper were observed, apparently forming a constituent of the hornblende of the rock. In this instance there seemed to be no doubt that the copper was derived from the lode. I should be much interested if any members of the Institute could give authentic instances of the occurrence of visible particles of gold actually forming part of crystalline or eruptive rocks at long distances from lodes.

Mr. Winslow's criticism|| raises the whole question as to the possibility of obtaining any experimental evidence, other than that founded on observation alone, with regard to the genesis of auriferous deposits, where the amount of metal to be looked

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* *Trans.*, xxvii., 997. † *Trans.*, xxvii., 597. ‡ *Trans.*, xxvii., 606.
§ *Trans.*, xxvii., 998. || *Trans.*, xxvii., 999.

for is extremely minute. This is a difficulty that must have presented itself to any honest worker who has made a study of the subject. The question is not so much whether the methods used by any particular investigator have been sufficiently refined, as whether any possible test may be delicate enough to give reliable results.

Now, while it is quite true that the results obtained by me do not show any difference between $\frac{1}{10}$ grain to the ton of country-rock (or one part in 156,800,000) and *nil*, and while, in my opinion, it is not possible to carry the refinement much further with safety, one must not, I think, lose sight of the fact that if the gold of any lode had been derived by segregation from the country-rock or from associated and subjacent rocks (it will be observed that lateral secretion is here indicated in its wider sense), such segregation must have been accompanied by local deposition at many points, as the auriferous solutions made their way to the lode-fissure. It is a matter of common observation that the country, even at considerable distances from lodes, is in many cases a network of miniature lodes, where deposition might reasonably be expected to take place. In collecting samples for analysis, care was taken to include many such places of possible deposition; yet the evidence obtained by me would almost justify the general statement that in solid country, where the possibility of impregnation from a lode or dike was reduced to a minimum, no trace of gold was present, even in the sulphides found. Numerous illustrations in support of this generalization are to be found in most of the tables, while the whole of the results of Table XII. bear on the subject.

The following errors, discovered too late to be corrected in Vol. XXVII., are here pointed out with the view of rendering the final version of my paper as nearly correct as possible. I believe that I shall find no more.

On p. 624, Table II., sample *m*, 6.7 grains should be 177.2 grains.

On p. 628, Table V., sample *g*, the yield of gold per ton of country-rock is correctly given as 1.8 grains. But an error in calculation made it, at first, 7.2 grains, and this is the yield plotted for that sample in Diagram 3. The diagram is therefore incorrect in this respect.

On p. 629, Table V., sample B, .0008 grain should be .008, and 0.4 grain should be 4.0.

On p. 636, Table IX., sample *h*, 4.5 grains should be 4.05.

On p. 637, Table X., it should be noted that samples *a* to *e* were taken from one level, and samples *f* to *k* from another. This accounts for the apparent discrepancy in the distances of the various samples from the reef. In the same table, sample *g*, 150 feet should be 140.

On p. 647, Table XVI., sample A1, 4.5 grains should be 4.05.

Mining Districts of Colombia.

Communications in Discussion of the Paper of Henry G. Granger and Edward B. Treville, Quibdo, Colombia. (See p. 33.)

(Atlantic City Meeting, February, 1898.)

ERNEST R. WOAKES, Cana, Colombia: The allusion in this paper, under the heading "Mining on the Isthmus," to the Espiritu Santo mine at Cana is neither according to fact nor worthy of the most interesting and able summary of the mining districts of Colombia contained in the remainder of the paper. The following remark is certainly open to contention:

> "The most widely-known mine of this department is the Espiritu Santo, at Cana, a quartz-mine opened up in 1680 and abandoned in 1727 on account of the attacks of the Cana Indians. This mine undoubtedly produced great quantities of gold, although many of the accounts given of its ancient richness are probably exaggerated. In 1884 it was rediscovered and reopened by the Darien Gold Mining Company, and under the able management of Mr. Thomas H. Leggett, now prominent in South African mining circles, produced for a time successful results; but later the veins became pockety and the output unsatisfactory."

The mine here referred to, though situated at Cana, was not the Espiritu Santo mine, and was closed down in 1892. Mr. Leggett, however, ran ten stamps on the ore from it for some two or three months in the year 1890, when he left Colombia. The Espiritu Santo mine was not reopened by the Darien Gold Mining Company until the year 1894. For the last three years the ore from the old Spanish workings of this mine has been yielding in the mill 1 ounce to the ton, and has served to keep the mine going whilst extensive development-works have been

carried on. These old workings were found to extend to a depth of nearly 200 feet, and are very heavily watered. A vertical engine-shaft has now been sunk 350 feet, and two crosscuts driven into the lode below these workings have proved that the very high-grade ore worked by the Spaniards continues in depth. It is entirely owing to the difficulties attending the instalment of a large hydraulic pumping-plant, to cope with the water met in the lower levels, that the mine is not now making large returns. Since starting the mill on the ore from the old workings in 1895, gold to the value of nearly £60,000 has been extracted; and we believe the mine to be on the point of becoming a large gold-producer. A working-force of between 300 and 400 natives and 20 white men is employed. The mine is equipped with a modern 20-stamp Fraser and Chalmers mill with Frue vanners. A cyanide-plant is being erected for the treatment of the concentrates. There is a large compressed-air plant at present working the temporary pumps; and a system of ditches, nearly 8 miles in length, provides motive power to 5 Pelton wheels and a turbine, which operate, besides the above-mentioned machinery, a saw-mill, machine-tools and electric-light plants. The company has its own steamer running to Panamá. From the above short description it will be admitted that the Espiritu Santo mine, as it at present exists, is at least entitled to a place among the active mines of Colombia.

I hope shortly to submit to the Institute a paper on this interesting old mine. In the meantime I would call the authors' attention to my paper entitled "Notes on the Espiritu Santo Mine at Cana, its Drainage and Recovery," published in the *Transactions of the Institution of Mining and Metallurgy*, London, vol. iii., part ii., 1894–1895.

FRANK OWEN, El Perú, Venezuela: Messrs. Granger and Treville deserve the thanks of all interested in Colombian mining for their able paper on the subject. With the exception of a report made to the British Foreign Office in 1894 by Mr. G. Jenner (at that time British Minister at Bogotá),* their contribution is, as far as I am aware, the first well-considered description of the mining districts of Colombia, taken as a whole. Having resided in Colombia for some years, I take pleasure in

* *Foreign Office Papers*, Miscellaneous Series, No. 331, June, 1894.

testifying to the general accuracy of their statements and conclusions. The subject is such a large one that, perhaps, I may be permitted, in no carping spirit, to offer a few corrections.

An interesting account of the Espiritu Santo mine at Cana was read before the London Institution of Mining and Metallurgy by Mr. Woakes,* who is still, I believe, in charge of the works, and, according to latest reports, has now attained a considerable measure of success. Instead of the veins being pockety, as described by Messrs. Granger and Treville, I am credibly informed that they have now a lode 60 feet wide of an average value of $12 gold per long ton. Mr. Woakes said in his paper:

> "This No. 3 cross-cut has therefore proved a lode more than 30 feet wide with rich portions on both north and south walls, and some 20 feet of good pay stuff between. The actual south wall has not been met in any cross-cut, so that the width of this extraordinary deposit is unknown."

The Colombian Quartz Mining Co. also is working gold-mines on the Isthmus of Panamá, at Santiago de Veraguas.

Messrs. Granger and Treville, I think, hardly do justice to the important mining interests of the department of Tolima by cursorily dismissing them in a few sentences. The Frias mine, to which they refer, is owned by the Tolima Mining Co., and, when silver was at a good price, was returning for years handsome profits, as much as $60,000 (gold) per month. They shipped on an average 220 long tons of concentrates monthly, running from 100 to 4000 ounces of silver per ton; and when I was in Colombia the Frias ore-sacks were a familiar sight to travelers on the Magdalena river steamers. The Colombian Hydraulic Co. and the Gravel Gold-Mines of Colombia have both been considerable producers of alluvial gold; and I understand that the *débris* question is now in a fair way of being more or less satisfactorily settled. An English company is working a gold-silver lode at Transito, near Ibague, extracting the silver by the hyposulphite and the gold by the cyanide process.

It may further be of interest to remark that the celebrated English civil engineer, Robert Stephenson (son of the still more

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* See remarks of Mr. Woakes, above.

famous George), was at one time engaged in mining at Santa Ana, in the Tolima.

I am surprised that Messrs. Granger and Treville, in their description of the mines of the department of Antioquia, do not devote more space to the Remedios district, which is at present by far the largest gold-producer in the republic. As I was connected for some years with the Frontino and Bolivia G. M. Co., to which they refer, I take the liberty of enlarging a little on their account of its mines. This company, like the original Marmato Co. (now Western Andes G. M. Co., Ltd.), was first formed in or about the year 1823, after the independence of Colombia had been declared, by some of the Englishmen who fought in the "Légion Britannica" on the patriot side. The present company was formed in 1864, reconstructed in 1885, and since 1888 has uninterruptedly paid dividends of from 15 to 40 per cent. The monthly output of bullion averages some 4000 to 4500 ounces. They have about 60 heads of Californian stamps dropping, as well as some 120 heads of wooden stamps of the native pattern. I fancy, by the way, that Messrs. Granger and Treville are mistaken in the statement that this company's gold passes through the hands of Messrs. Ospina Bros., of Medellin.

The Frontino and Bolivia Co. also holds the controlling interest in the Antioquia (Frontino) Co., operating gold-mines in the Cerro del Frontino, and forms to-day much the most profitable and (with the exception of the St. John Del Rey G. M. Co. in Brazil) the oldest and largest gold-mining enterprise in South America.

The Colombian Mines Corporation has been working with success, for some years past, the Sucre, Providencia and other mines near Remedios.

The Franco-Belgian Co., *La Société des Travaux Miniers*, was working in my time (1891–4) the Cristales and San Nicolas mines at Segovia, near Remedios; but I hear they are not doing anything at present. They had a 30-stamp mill built by Fraser and Chalmers. For a fuller description of the Remedios mines, I may be allowed to refer to my own paper "On the Gold-Mines of the Remedios District," read before the Institution of Mining and Metallurgy, London, on the 16th October, 1895.*

* *Trans. Inst. M. M.*, vol. iv.

Several very fine specimens of wire-gold from the Santa Isabel mine at El Coco (distant about 15 miles from Remedios) were exhibited by the Colombian Government at the Chicago Exposition of 1893; and a specimen, formerly in my possession, is now to be seen in the mineralogical collection of the British Museum (Natural History), at South Kensington, London.

I have seen the unique collection of ancient Indian gold ornaments, etc., belonging to Don Leocadio Arango, of Medellin, to which Messrs. Granger and Treville refer. It is, I believe, valued at $40,000 (gold). A portion of the collection was exhibited at Chicago in 1893.

Zaragoza, on the river Nechí, is very rich in alluvial gold, although so far all attempts at dredging have failed there. The locally celebrated "Cristo de Zaragoza" has a crown of thorns and other gold ornaments to the weight of about 12 pounds (Troy). In the cathedral at Sta. Fé de Antioquia there is an image of the Virgin, of heroic size (say 7 feet high), made of solid silver.

An attempt was made, a few years ago, to reopen the old mines of the Spaniards at Guamoco, in the department of Bolivar, by Col. J. H. Dunstan, a well-known mining man of Roanoke, Va., but I believe the difficulties of transport proved too great.

The pertinent remarks of Messrs. Granger and Treville as to the government and people of Colombia, and their fair and cordial treatment of well-behaved foreigners, will be fully borne out and appreciated by those who are in a position to compare that country with some of the other Hispano-American republics, where, to misquote the old proverb, "Honesty breeds contempt." Whether it is owing to the strong element of Jewish ancestry (due to the Jews driven out of Spain in the time of Ferdinand and Isabella) or to the more temperate climate, the inhabitants of Antioquia are, so far as my experience goes, not only amongst the best miners, but the most hardworking and reliable people generally in South America. The comparative stability of the government and the great richness and variety of the mineral resources offer considerable guarantees, to those investing in well-planned and well-directed mining enterprises in Colombia, that they will not

"See their ship of glass
Wrecked on a reef of visionary gold."

Much could be written on the fascinating subject of mining in Colombia in the time of the "conquistadores." Don Vicente Restrepo's book, *Estudio sobre las Minas de Oro y Plata de Colombia* (of which an English translation by C. W. Fisher was published at New York in 1886), contains much interesting and, on the whole, trustworthy information on this point.

Kalgoorlie, Western Australia, and its Surroundings.

Discussion of the Paper of George J. Bancroft, Denver, Col. (See p. 88.)

(Atlantic City Meeting, February, 1898.)

EDWARD S. SIMPSON, Perth, Western Australia (communication to the Secretary): In 1896 the East Coolgardie gold-field was divided into three fields, viz.:

	Area, square miles.	Official centers.
(1) East Coolgardie, . .	634	Kalgoorlie.
(2) Broad Arrow, . .	588	Kurawa.
(3) Northeast Coolgardie, .	22,960	Kanowna.

The whole of the area embraced by the eastern gold-fields is covered with late Tertiary sands and gravel, with occasional beds of clay shales. These mostly overlay granite which occasionally shows through in small round hills. At Coolgardie, Kalgoorlie and other mining centers, belts of diorite, serpentine and highly metamorphosed sedimentary rocks appear on the surface.

The rainfall at Kalgoorlie in 1897 was 4.75 inches. That the climate has not always been so arid is evidenced by the recent discovery at Coolgardie (only twenty-five miles away) in the late Tertiary formations of a thick bed of brown coal, containing numerous fossil fern-leaves.

As assayer and mineralogist to the Government of Western Australia I have had occasion to examine a large amount of the Kalgoorlie ores, both in Perth and at the mines at Boulder itself. The only tellurides I have been able to determine are

calaverite and coloradoite, both of which frequently occur in large masses in these mines. The following are analyses by me of these minerals:

	Calaverite.	Coloradoite.
Tellurium,	57.27	[49 48]
Gold,	41.37	Trace.
Silver,	.58	.12
Mercury,	....	50.40
	99.22	100.00
Specific gravity,	9.311	8.068

The auriferous "lodes" in all the mines at the south end of the camp (Boulder) are in reality dikes of highly foliated felstone, impregnated with carbonates of lime, etc., and also with auriferous pyrites and tellurides of the noble metals. At the surface this has been leached and converted into a ferruginous kaolin, but in depth the true character of the rock is very evident. An analysis of typical lode-stuff from the 300-foot level of the Lake View Consols mine gave the writer the following result:

Lake View Schist.

Water H_2O.	Hygroscopic,	.402
	Combined,	1.809
Soluble in hydrochloric acid.	Calcium carbonate, $CaCO_3$,	10.882
	Magnesium carbonate, $MgCO_3$,	6.315
	Ferrous carbonate, $FeCO_3$,	1.553
	Ferrous oxide, FeO,	1.360
	Ferric oxide, Fe_2O_3,	1.541
	Alumina, Al_2O_3,	1.326
	Manganese oxide, MnO,	Trace.
	Phosphoric oxide, P_2O_5,	Trace.
Soluble in nitric acid.	Iron, Fe,	3.990
	Sulphur, S,	4 417
	Tellurium, Te,	Trace.
Insoluble.	Silica, SiO_2,	51 271
	Titanic oxide, TiO_2,	.226
	Alumina, Al_2O_3,	12.519
	Ferrous oxide, FeO,	.311
	Lime, CaO,	.313
	Magnesia, MgO,	1.159
	Undetermined and loss,	.606
		100.000

Gold, 9 oz., 12 dwt., 18 gr. per ton.
Silver, 6 oz., 7 dwt., 8 gr. per ton.

Contrary to Mr. Bancroft's statement, one would naturally

expect that these dikes would extend to unlimited depths, and up to the present there is not a single instance of their having pinched out in depth. What Mr. Bancroft regards as cross-courses are merely branches of the dikes, and, not unnaturally, no enrichment of the dikes is noticeable at the point where they fork.

That not all the miners on this field are sluggards is proved by the fact that the main shaft at the Associated gold-mines, which is 12 feet 6 inches by 4 feet 6 inches in section, was sunk last week no less than 32 feet through hard diorite.

When the writer visited Boulder at Easter of this year (1898), two of the large mines were dealing successfully with their slimes by means of filter-presses. At Hannan's Brownhill, the ore is crushed dry and then divided by a pneumatic separator into sands and slimes. The former are cyanided in vats. The latter are agitated for six hours in a vat with a 0.3 per cent. cyanide solution, and the vat-pulp is forced into a filter-press. The gold-bearing solution is thus removed from the ore, which is washed by forcing water through the press. The whole operation of filling the press, leaching and emptying takes one and one-half to two hours.

At the Lake View Consols, the wet tailings from the battery are partially settled, and the slimy liquor is treated in a set of 10 filter-presses. These latter are working very satisfactorily, giving 85 per cent. extractions on 10 dwt. slimes. Each press can treat 1 ton of slimes per hour.

Sulphide works are in course of erection at several of the mines and will deal with all the unoxidized ore from them. The ore will be crushed dry, roasted in mechanical furnaces and leached in vats with cyanide.

The Coolgardie water-scheme is now well in hand, and should be completed in about three years, when 5,000,000 gallons will be pumped to these fields daily.

The output of the Kalgoorlie field since 1895 has been as follows:

	Bullion. oz.	Approx. value.
1895,	36,960	$665,280
1896,	103,171	1,857,078
1897,	302,342	5,431,956
1898 to 30th April,	123,557	2,224,026
Total,	566,030	$10,178,340

R. H. Terhune, Salt Lake City, Utah: It is to be regretted that Mr. Bancroft has not given us a more detailed description of the process of "dry blowing." The subject is one of interest and, in many American districts, of great possible importance.

Mr. Bancroft: The process of dry blowing is employed in West Australia by reason of the lack of water available to the alluvial mines. The wind is generally blowing on the gold-fields, and the prospector takes advantage of this fact to aid him in getting the gold from the dry soil. On a still day dry blowing is not only very unsatisfactory, but very unpleasant as well. Even on a breezy day, it seems almost impossible to keep one's head out of the dense cloud of dust which rises from a dry blower.

The machines used are shown in Figs. 4 and 5 of my paper. The "dry blower" is simply a box, containing slanting shelves, made of screens of different mesh. The material is sized by the screens and concentrated behind riffles. That is all there is of the process, as far as the dry blower is concerned. The wind is useful in carrying off the fine stuff throughout the process.

The box is mounted on four legs made of slender sticks, which are nailed fast to the box. The lower ends are sometimes nailed to cross-pieces and sometimes not. In either case the ends are buried in dirt to prevent the machine from travelling. One of the operators grasps the side of the box with both hands and shakes it back and forth at the rate of about forty double strokes per minute. The springy legs enable this to be done without the necessity of having any complicated joints, as would be necessary if the legs were rigid. The mate of the man who shakes the table shovels dirt into the apparatus. A machine 2 feet by 3 feet in size will keep a man shoveling constantly, but not energetically. I regret that I cannot give the capacity in pounds per hour. Most of the diggers make their own apparatus; hence one sees all kinds of machines and all possible arrangements of screens.

There are a few factory-made machines in use, with a bellows attachment beneath the screens, so arranged that the air is forced through the screens by the shaking motion. The dig-

gers say these are "no good"; and no doubt they are right, although the same principle is used in "Wood's dry placer-miner," with great success.

After a certain amount of material has been handled, the portion lodged behind the riffles is scraped out and treated by pan. The pans are like ordinary gold-pans, except that they are rounded where the rim joins the bottom, like a wash-basin. Some are made of iron and some of wood.

To pan the concentrates, the material is poured from one pan to another, giving the wind a chance to carry away the lighter part. One pan is placed on the ground while the material is poured into it from the other held at a height of 4 or 5 feet. Then the sand is tossed in the air from a pan and caught again, with the wind acting as a concentrating agent. Finally, the digger uses his breath to blow away the light stuff. The particles of gold are then picked out by hand, or gathered by mercury.

It is surprising what fine gold can be saved. I was told that even with abundant water it would not pay to work over the tailings from a dry blower. Coarse nuggets whenever they find their way into the machine, are readily picked out from the first riffles by hand. To make the dry blowing method successful the alluvium must be perfectly dry and loose. Clay will not work well, for the cakes of clay carry enclosed particles of gold over the riffles.

For working large quantities of alluvium, a machine manufactured by Fraser & Chalmers, called "Wood's dry placer-miner," has proved very effective. The machine is equipped with shaking-screens and a pair of bellows which force air back and forth through the screens, as a jig does water. A man who had run one told me that it would save gold as fine as mustard.

Notes on the Stockholm Exposition and the Iron and Steel Trade of Sweden.

Discussion of the Paper of James Douglas, New York City. (See p. 101.)

(Atlantic City Meeting, February, 1898.)

CHARLES H. MORGAN, Worcester, Mass. (communication to the Secretary): In connection with Mr. Douglas's mention of the continuous charcoal kiln used at Kopparberg, some further data concerning this important invention may be of interest. Before giving these, however, I will say a word or two concerning other features of Swedish practice.

In 1894 I spent two days at Soderfors with Mr. Tom Bergendal, the manager of the works, and was specially interested in the gas-producers and wood-drying kiln, illustrated in the paper contributed to our *Transactions* by Mr. Odelstjerna in that year.* The wood-drying kiln is, however, quite different in design and construction from Ljungberg's charcoal kiln.

In the summer of 1894 it was discovered that by placing the gas-producer as close as possible to the open-hearth furnace, and working the producer *hot*, ordinary weather-dried wood, still containing a considerable percentage of moisture, could be used to advantage in the melting-furnace. Hence, the drying-kiln was needed for wet wood only.

One of the iron-works of the Udeholm Co. was the first to adopt the Lundin saw-dust gas-producer and condenser, enabling them to make gas from saw-dust containing 45 per cent. of moisture, and convey the gas from the condenser through wooden pipes to the Siemens steel-melting furnace.† This saw-dust gas gave very satisfactory results in iron-works; but the proprietors experienced much trouble by reason of the pollution of rivers by the tar-water from the condenser.

On my visit, in 1894, I found that the Udeholm Co. had put in for their steel-melting furnaces the same wood-gas producers

* *Trans.*, xxiv., 288. Figs. 17 to 21, inclusive.

† See Hon. A. S. Hewitt's *Report of the Paris Exposition of* 1867.

as those used at Soderfers. Mr. Janssen, the manager, emphasized the importance of placing the producer as close as possible to the furnace, so as to introduce the gas *hot.*

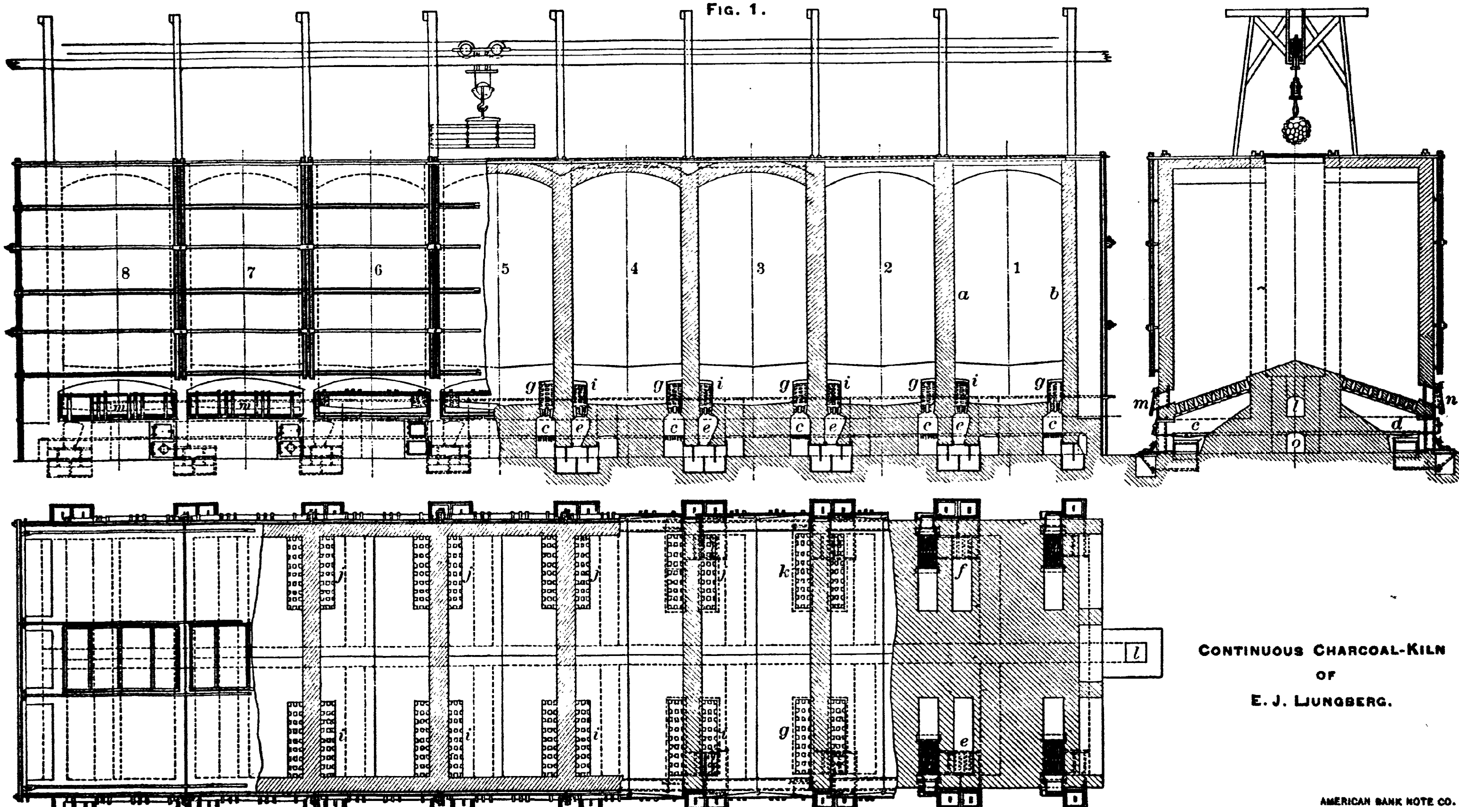
This change of practice is interesting. Thirty years ago it was considered necessary to place Siemens furnaces using gas from bituminous coal at a long distance from gas-producers, and to employ iron pipes exposed to the air, so as to cool down the gas. But most iron-masters now prefer to place the producer close to the furnace, so that the tar may be utilized in the latter, instead of clogging the flues.

Returning to the Ljungberg charcoal-kiln, I offer the following brief description:

These kilns, and the accompanying appliances for the mechanical handling of wood and charcoal, were invented by Mr. E. J. Ljungberg, the President and managing director of the Stora Kopparberg Co. The kilns are arranged in series of four chambers, built together, having flues in common, and so arranged with water-sealing valves that each chamber may be connected or disconnected with any other of the series, or with the common flue leading to a chimney or an exhausting-fan. Fig. 1 (taken from *Jernkontorets Annaler*, 1897) shows plan and sections of a plant comprising eight chambers, or two such series, numbered from 1 to 8. Wood is charged of the length corresponding to *a b* (No. 1), the width of the shaft. Each chamber is provided with the grates *c* and *d*, gas-escapes *e* and *f*, and discharge-holes *m* and *n*. From the grate, the products of combustion pass through a number of smaller holes, *g* and *k*, over the mass of wood, and out at *i* and *j*.

The operation is carried on in each set of four chambers in four stages: preheating and partial drying, complete drying and partial charring, complete charring, and final quenching and withdrawing. The contents of each chamber pass through these four stages; and at any given moment each chamber is in one of the four. Thus, for example, No. 1 is charged with wet wood, taken directly from a river or pond, and this is partially dried by hot steam and gases from No. 2, which has already passed through that stage, and in which incipient charring is now going on. No. 2 in turn receives the still hotter gases of No. 3, where the charring is being completed; and finally No. 4 is shut off from the rest, having passed through

Fig. 1.



Continuous Charcoal-Kiln
of
E. J. Ljungberg.

American Bank Note Co. N.Y.

all stages except that of quenching and discharging. When No. 4 has been discharged and re-charged, it will be reconnected with the series, so as to receive from No. 1 (by this time in its second stage) the hot steam and gases for preheating and partial drying. The order then becomes 4, 1, 2, 3; No. 3 is shut off, to be quenched and discharged; and so on, in continuous operation. The entire cycle requires 20 days. The charcoal drawn from the kiln is carried by an aerial tramway directly to a blast-furnace or to a storehouse.

The by-products per cubic meter of wood charged are: 2.76 kilogrammes of tar, 2.69 kilogrammes of acetic acid, and 0.96 kilogramme of methyl alcohol. Calculated per cord of wood, this would be 22.1 pounds of tar, 21.47 pounds of acetic acid, and 7.65 pounds of methyl alcohol.

The great national importance of this improvement may be inferred from the fact that its operation for four years at Domnarfvet has proved the cost of operation to be one-third that of burning in ordinary heaps,* while the yield of charcoal is 22 per cent. greater. The saving of by-products is an additional gain. But the most striking and obvious advantage is the practical increase by nearly one-fourth of the charcoal-resources of Sweden—or, what is the same thing, the reduction by that amount of the quantity of forest-lands necessary to maintain a given supply of this national metallurgical fuel.

The system should be of great value to charcoal blast-furnaces located (like some in the Lake Superior region) near the ore-supply, and so situated that the wood could be floated to the kilns and furnaces.

The Elimination of Impurities from Copper-Mattes in the Reverberatory and Converter.

Discussion of the Paper of Edward Keller, Baltimore, Md. (See p. 127.)

(Atlantic City Meeting, February, 1898.)

E. D. Peters, Jr., Dorchester, Mass.: This paper of Mr. Keller's seems to me a step in a direction that has been very little exploited, and is likely to lead to valuable practical re-

* This includes the saving effected by the mechanical arrangements for handling wood and charcoal.

sults. There are already in existence a great number of reliable and exhaustive analyses of almost every conceivable product and educt in connection with copper-metallurgy. By delving into the archives and transactions of the various scientific societies of the past fifty years, and especially into German technical literature, analyses of almost any desired substance connected with copper-smelting can be found, from ore to refined copper, and from slag to flue-dust.

These determinations possess considerable value as a means of increasing our knowledge of the intimate composition of all these materials, and also as an indispensable aid in perfecting the chemical methods by which these minute proportions of foreign metals and metalloids can be quickly and accurately determined.

But as a means of throwing any positive light upon the exact stage and proportion in which these impurities are eliminated during the smelting-processes, or of furnishing any guide to the practical metallurgist in making better copper, they are, for the most part, a distinct failure. The reason why so much exact and laborious work has produced such slight practical results is obvious. It is because most of the determinations hitherto made have dealt with isolated products, and have represented but small quantities of material.

Mr. Keller has had unprecedentedly large amounts of carefully sampled matte and other products at his disposal, and has utilized them with the skill of the chemist, backed by much of the practical sagacity of the metallurgist. His material consisted of the high-grade copper-matte produced in such enormous quantities by the Anaconda Copper Company of Montana, averaging about 61 per cent. of copper, and carrying about 1 ounce of silver to 1 per cent. of copper. The matte contained also about 0.2 ounce gold to the ton, and, besides the proper proportions of sulphur and iron, carried appreciable amounts of zinc, lead, bismuth, antimony, arsenic, selenium, tellurium and magnetic oxide of iron. Excepting nickel and cobalt, we have here almost every substance likely to occur in the smelting of ordinary copper-silver-gold ores. Mr. Keller's opportunities were made doubly valuable by the fact that the matte-product of the Anaconda was divided, and treated by two different and rival methods—the regular Swansea reverberatory

system and the Bessemer converter method—the old and the new. (Since 1895 the former method has been discarded.) Thus he was in a position to make a parallel series of determinations on all the products and educts of the two great matte refining processes, and on material which, for quantity and evenness of quality, could not be duplicated elsewhere in the world.

The subject is so complicated, the number and variety of determinations are so great, and the conclusions arrived at are so dependent upon an accurate understanding of the particular conditions corresponding to the various samples taken, that it would be useless to attempt to generalize about them in a brief discussion of this paper. Even Mr. Keller has but sparingly attempted to do so, leaving only his plain facts and results on record as a landmark for those who will doubtless follow in the same path, and enlarge the fruitful territory that he has so industriously opened.

There is one point, however, in which Mr. Keller's laborious work is a little disappointing to the practical smelter. This is the fact that he touches but lightly upon the loss of silver during the concentration of matte and refined copper, paying to this valuable metal no more attention than to bismuth or tellurium. Of course, from a scientific standpoint, silver is not more important than bismuth, but practically and commercially it is many times more important; and, as Mr. Keller would have little opportunity for the pursuit of minute proportions of either bismuth or tellurium, if the silver were not present to pay for it, it would seem only fair that this valuable element should have a trifle more recognition than its exact scientific quota. Such results, however, as Mr. Keller gives us, appear to confirm the opinions already held by practical metallurgists on the subject, viz., that there is a very appreciable loss of silver in the converting of argentiferous copper-matte, and that this loss is mainly referable to volatilization. To this I will add my own opinion, and that held, I think, by most metallurgists:

1. That, other things being equal, the loss of silver is greater in the presence of certain volatile metals and metalloids, such as zinc, lead, bismuth, antimony, arsenic, tellurium, etc.

2. That the loss of silver increases with the pressure of the blast, and the violence of the oxidizing reactions.

3. That the loss of silver becomes greater as the matte grows richer in the converter, reaching its maximum when the sulphur and iron have been mostly eliminated. It is on account of this heavy loss of silver toward the close of the converter-process that I have suggested a trial of the direct refining method, as practiced by Mr. Nicholls at the works of the Cape Copper Company, Ltd., at Britonferry, Wales. As many of the members doubtless recollect, this process consists in utilizing the reaction that occurs between the oxides and sulphides of copper when melted together in proper proportions. All the copper present is changed to the metallic condition, while the sulphur is entirely eliminated as SO_2. While I am not aware that this method has ever been used for the treatment of highly argentiferous white metal, it yet seems reasonable to suppose that the loss of silver by volatilization would be considerably less in a reverberatory furnace than in the converter.

A. R. LEDOUX, New York City: I would like to ask Dr. Peters whether they are working that process now on argentiferous mattes and on impure mattes. I was in Britonferry two years ago, and they were handling there Newfoundland mattes containing a certain amount of silver, with comparatively little lead, zinc, and other impurities; and it has always been a question in my mind whether they would succeed in treating by that method some of the impure mattes that we have to fight with, in Montana, for instance.

DR. PETERS: No; they were not treating any of the mattes which Dr. Ledoux speaks of, because they had all that they could handle from the Cape and from Newfoundland, but at one time they had also one furnace running on mattes containing considerable silver and lead, and especially antimony and arsenic. As far as antimony and arsenic went, the elimination was very satisfactory; but as to the lead, which I think amounted to 4 or 5 per cent. in the mattes, it was otherwise.

WILLIAM KENT, New York City: I notice, in looking over Mr. Keller's paper, that the percentages of antimony left in the refined copper are considerably higher than those given in Mr. Sperry's paper on "The Influence of Antimony on the Cold-Shortness of Brass" (*ante*, p. 176), as the allowable percentage of that element for the manufacture of brass.

It would be interesting to know whether the process of which Dr. Peters speaks will produce copper pure enough for that manufacture, so that the trade will not be dependent on Lake copper, as I believe it now is.

R. P. ROTHWELL, New York City: I have recently received a letter from the manager of the Britonferry works, reporting that the process is still in practical commercial operation, with the same economical results as those which Dr. Peters observed and reported in 1895.*

DR. LEDOUX: I have received within a week a letter saying that they are applying the process to varied material, and are so well satisfied with its success that they are taking steps to introduce it in the United States.

MR. KELLER (communication to the Secretary): In reply to the remarks of Dr. Peters, expressing disappointment at the slight treatment, in my paper, of the commercially very important loss of silver, I would say that this neglect was at least partially deliberate on my part, and that such a criticism was not unexpected.

To treat this subject thoroughly would involve questions of loss and gain which smelting companies generally are loath to have discussed, and it was not my intention to give to my paper any commercial aspect.

On the other hand, the investigation of this part of the subject was cut short by technical difficulties.

Errors of a few per cent. on such small amounts of bismuth, tellurium, etc., as are present in the material treated, though unavoidable in the sampling and analysis, would still leave the conclusions reached as valuable, for the purpose of metallurgical guidance, as if the analytical results were absolutely accurate. This is not the case with silver. By reason of its commercial value, a deviation of a few per cent. from absolute accuracy in determination of this metal would be highly unsatisfactory to myself, and perhaps highly misleading to others. I therefore declined to take the responsibility for such errors.

I cannot quite agree with the opinion of Dr. Peters that the

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* *Modern Copper Smelting*, 7th ed., 1895, Sci. Pub. Co., N. Y., p. 519 *et seq.*

so-called direct method of copper production should, at least partially, supersede the converter.

It is, after all, but a limited improvement of the old reverberatory process.

Some years ago it was tried at the Baltimore Copper Works in the working of Anaconda mattes, but was not adopted, because it showed no marked advantage over the old method.

In large smelting-works especially, the direct method does not seem to recommend itself to the metallurgist, because it reintroduces greater length of time to produce copper, repeated handling of the material, and consequent complication of the whole system. In short, it presents no economical advantage.

The main advantage pointed out by Dr. Peters is the reduced loss of silver by volatilization. Such a loss in a converter-plant would naturally be guarded against by the construction of adequate dust-chambers, thus only slightly increasing the first investment.

The author of the direct process, Mr. Christopher James, has described it in a pamphlet in the most glowing terms as the method of the future. Among his more remarkable claims is the assertion that there is *no* loss, and that the amount of copper produced is greater even than the amount called for by the humid assay of the matte.

To me, the best feature in this process seems to be that it permits the production of undiluted sulphur dioxide; a result not attainable in ordinary smelting and calcining. When fusion by exterior heat is once started, the reaction of the sulphides and oxides continues of itself, and thus produces a comparatively pure sulphur dioxide, unmixed with atmospheric or furnace-gases.

For the purpose of sulphuric acid manufacture, the process might, therefore, well be introduced at some works as an auxiliary.

As already observed, the direct process necessitates repeated handling and transfer of material, with consequent large waste of heat. The converter-process may be made continuous, since waste of heat is almost entirely avoidable. The molten matte can be transferred at small cost from the matting-furnace to the converter; and from the latter, the molten copper can be conveyed with equal facility to a refining-furnace of any kind. Up

to the present day this has not been accomplished at any one of the smelting-works in the United States. None of them have been harmoniously designed, with the ultimate purpose of making finished copper from the matte.

At the Parrot smelter, in Butte City, during the administration of Mr. H. A. Keller, the converter was brought to a high state of efficiency through improved construction, mechanical devices, and method of lining. The practice of chilling the matte and remelting it for the converter was continued at this smelter. To transfer the molten matte from the matting-furnaces to the converters was a progressive step first taken at the Boston and Montana Company's Great Falls works, and followed by the Montana Ore Purchasing Company in Butte City, under the administration of Mr. D. E. Heller. It has been reported that the last step in a continuous converter-process, that of transferring the molten copper from the converters to a refining-furnace, has lately been adopted at the Anaconda Company's works.

I have especially appreciated Dr. Peters' suggestions, as they have induced me to study, to some small extent, in an experimental way, on a laboratory basis, the elimination of impurities from copper-mattes by the direct method, and thus to show what quality of copper that method, as compared with the other methods, is most likely to produce. This difference in product I believe to be clearly demonstrated by my investigation. Since the direct process proper is based strictly on chemical reactions, no mechanical operations playing any part in it, there appears to be no reason to assume that the results obtained in the laboratory crucible are not exact parallels to those obtained in the reverberatory furnace. I subjoin the results of my experiments as a supplement to my original paper.

The Elimination of Impurities from Copper-Mattes in the Direct Process.—The energy of a chemical reaction between two elements is measured by the amount of heat evolved, expressed in calories, in the process of their combination; and there is a thermo-chemical law that, in general, in the presence of more than two elements those two will combine which by their combination will evolve the greatest number of calories. To decompose the combination of two elements, the same amount of energy similarly expressed in calories is required as

was evolved by their union. These laws govern the reactions in our metallurgical processes.

The chemical nature of the process under discussion may be expressed in one or both of the following equations:

$$\text{I. } 2\,Cu_2O + Cu_2S = 6\,Cu + SO_2.$$
$$\text{II. } 2\,CuO + Cu_2S = 4\,Cu + SO_2.$$

The fact that the heat evolved by the formation of SO_2 is greater than that evolved in the formation of the sulphides and oxides of copper is the thermo-chemical explanation of these reactions.

In the converter the reaction, in the simplest form, finds expression in the equation:

$$Cu_2S + 2\,O = 2\,Cu + SO_2.$$

Here the full energy of the free oxygen is available for the purpose of oxidizing impurities. In the direct process a part of that energy is consumed in the decomposition of the copper oxides; *i.e.*, in the liberation of the oxygen. This is, roughly stated, the difference in effectiveness between free and combined oxygen. We therefore expect the converter to produce the purer copper. Let us examine the results of the direct process.

Arsenic.—In these experiments copper-mattes and copper oxides were used, and the fusions were conducted in covered 10-gramme crucibles. When slags were fused for the determination of copper or impurities the fusion was made in the original crucible, to avoid loss by slag adhering to the latter.

The matte and impure oxide employed for my arsenic and antimony determination had the following contents:

	Arsenic. Per cent.	Antimony. Per cent.
Matte,	0.058	0.097
Copper oxide,	0.049	0.087

First Fusion.—Weighed in:

	Grammes.		Arsenic. Gramme.
Matte,	20,	containing	0.0116
Pure copper oxide,	25,	"	0.0

Found:

	Arsenic. Gramme.
In metallic button,	0.0095
In slag,	0.00171

The arsenic in the button is in round numbers 82 per cent. of the total arsenic; the (apparent) elimination, therefore, is 18 per cent.

The slag obtained by this simple fusion was, however, a tough magnetic mass, containing much metallic copper, which in practice would not be permissible. It was for this reason deemed proper to determine the copper in the slag and to apportion to it the same quantity of arsenic as was found in the button.

The results thus obtained are as follows:

	Copper. Grammes.	Arsenic. Gramme.
In metallic button,	26	0.00950
In slag,	4.5	0.00171
Apportioned to copper in slag,		0.00164
Remaining in slag proper,		0.00007

From these data it follows that the copper produced retains 97 per cent. of the arsenic, and that the true elimination is only 3 per cent. This small figure falls well within the limit of errors, and it may be that the elimination is somewhat greater; or we may assume, with an equal degree of probability, that there is no elimination at all.

Second Fusion.—Weighed in:

	Grammes.		Arsenic. Gramme.
Matte,	20,	containing	0.0116
Copper oxide,	25,	"	0.0123
Total,			0.0239

Found:

	Copper. Grammes.		Arsenic. Gramme.
In metallic button,	23,	containing	0.0165
In slag,	9.5,	"	0.00632
Apportioned to copper in slag,			0.00684

Here the apparent elimination of arsenic is 31 per cent., while, when the proper amount is apportioned to the copper contained in the slag, the elimination is again but 3 per cent.

Third Fusion.

The resulting buttons of the first two fusions had a small admixture of sulphur. The amount of copper oxide in the third fusion was, therefore, increased; and to avoid metallics in the slag the mixture in the crucible was covered with borax-glass in sufficient quantity to insure a fusible slag.

Weighed in:

	Grammes.		Arsenic. Gramme.
Matte,	20,	containing	0.0116
Copper oxide,	30,	"	0.0147
Total,			0.0263

Found:

	Copper. Grammes.		Arsenic. Gramme.
In metallic button,	32,	containing	0.0220
In slag,	3.5,	"	0.00331
Apportioned to copper in slag,			0.00245

The arsenic retained in the button is in round numbers 84 per cent., or the elimination is 16 per cent; but when, in this case also, as in the previous fusions, the large amount of copper in the slag is considered, the elimination is reduced to 7 per cent. The increased amount of copper oxide, therefore, does not materially aid the elimination of arsenic.

Antimony.—The recital of details for this element would be a repetition of what has been said about arsenic. The fusions of 20 grammes of matte with quantities of 25 and 30 grammes of copper oxide gave results showing an elimination of antimony varying between 50 per cent. and 75 per cent. Making allowance for the copper in the slags, the true elimination of antimony is probably, on the average, close to 60 per cent.

Selenium and Tellurium.—Repeated fusions showed, to my surprise, that selenium is almost completely eliminated by this method. Even after adding as much as 0.1 gramme of selenium to the matte, only traces could be found in the buttons and slags resulting from the fusion.

Tellurium, on the other hand, shows no elimination. It could not be detected in the slags, while in the buttons the total original amount of tellurium was quantitatively determined.

Bismuth.—For the purpose of ascertaining the behavior of this metal, two fusions were made.

First Fusion.—Weighed in:

	Grammes.		Bismuth. Gramme.
Matte,	20,	containing	0.11
Copper oxide,	25,	"	0.0

Found:

	Bismuth. Gramme.
In metallic button,	0.066

Or 60 per cent., showing an elimination of 40 per cent.

Second Fusion.—Weighed in:

	Grammes.		Bismuth. Gramme.
Matte,	20,	containing	0.11
Copper oxide,	30,	"	0.0

Found:

	Bismuth. Gramme.
In metallic button,	0.075

Or 68 per cent., showing an elimination of 32 per cent.

The slags were not tested, nor was the copper therein taken into consideration.

Lead.—A matte containing 3.025 per cent. of lead was fused with pure copper oxide.

First Fusion.—Weighed in:

	Grammes.		Lead. Gramme.
Matte,	10,	containing	0.3205
Copper oxide,	15,	"	0.0

Found:

	Lead. Gramme.
In metallic button,	0.0182
In slag,	0.2040

From these figures the distribution of the lead after fusion is found to be as follows:

	Per cent.
In metallic button,	6
In slag,	67
Volatilized,	27
Total elimination,	94

Almost identical results were obtained with a more distinctly leady matte, containing 18.54 per cent. of the metal.

Second Fusion.—Weighed in:

	Grammes.		Lead. Grammes.
Matte,	10,	containing	1.854
Copper oxide,	15,	"	0.0

Found:

	Lead. Grammes.
In metallic button,	0.0929
In slag,	1.2391

Calculated from these results, the lead is distributed as follows:

	Per cent.
In metallic button,	5
In slag,	67
Volatilized,	28
Total elimination,	95

The buttons from the above-described fusions contained no sulphur in appreciable quantity. In one instance, however, when sulphur was present to a very noticeable extent, the elimination of lead was found to be lowered to 84 per cent.

From the foregoing results we learn that the impurities in the mattes, as effected by the reaction of the sulphides and oxides of copper in fusion, can be classed in three groups. The first, forming one extreme, comprises tellurium and arsenic, which show practically no elimination. The second, forming the other extreme, comprises selenium and lead, showing almost total elimination. The third group stands between the other two, with bismuth nearer the former, and antimony closer to the latter.

The copper thus produced does not seem to excel in purity that of the reverberatory, and does not nearly approach that of the converter.

The grouping of the elements, as above given, is entirely due to the relative affinities of those which take part in the reaction. The evolution of sulphur dioxide has, in my opinion, no influence on the elimination of the other elements. We do not know how some of these are combined before the fusion; but, judging by results after fusion, we may regard it as most probable that the members of the first group form an arsenide and a telluride of copper, which do not enter into reaction with the oxides of copper as do the sulphides of the latter metal.

In the second group selenium and lead do not combine with copper. They, like the sulphur, are oxidized by the oxygen of the copper oxide. The reaction for lead, which is undoubtedly present as sulphide, is expressed by the equation:

$$PbS + 3\,CuO = 3\,Cu + PbO + SO_2.$$

For the third group it is evident that the reactions of both of the other groups must take place in almost equal proportion, or that the affinity of bismuth and antimony for metallic copper and for the combined oxygen in the copper oxides is about equal.

In reply to Mr. William Kent's observation, that the refined copper, often referred to in my paper, contains more antimony than is permissible for the production of good brass, I wish to point out that most of the copper spoken of as refined copper is argentiferous copper, which is not put on the market for the manufacture of brass, etc., but is merely refined as a preparation for electrolysis. To conform to the standard as set forth by Mr. Erwin S. Sperry, in his paper on "The Influence of Antimony on the Cold-Shortness of Brass" (*ante*, p. 176), electrolytic copper is required. This is now produced in enormous quantities, equal and superior in quality to Lake copper, from comparatively impure, argentiferous converter-copper.

On page 14 of my paper may be found the results of an analysis of good electrolytic wire-bar copper (*ante*, p. 140).

Mr. Keller (a later communication to the Secretary): In my reply, in a former part of this discussion, to the remarks of Dr. Peters on my paper, I said that the transferring of the molten matte from the matting-furnace to the converters was a progressive step first taken at the Boston and Montana Company's Great Falls works, and followed by the Montana Ore Purchasing Company in Butte City, under the administration of Mr. D. E. Heller. The statement that this improvement was made under Mr. Heller's administration was an error, for which I am alone responsible. As I learn from Mr. Heller, the idea of the method in use at the works in question was conceived by Mr. F. A. Heinze, President and General Manager

of the company, and it was installed during the administration of Mr. H. C. Bellinger.

The accompanying views, which Mr. Heller has kindly furnished, convey a clear idea of the operation referred to. In Fig. 1 two converters are seen, both in operation, one being blown and the other being skimmed of the slag. On the latter, to the right, are noticeable two pins, to which, on either side of the converter, the four chain-ends, suspended from a pulley-block and traveling-crane above, are attached. The converter, when ready for filling, is lifted from its stand and carried to a pit at the matting-furnace, as may be seen in Fig. 2. The pit is behind and a little to the left of the group of men. The converter is lowered into this pit, and a spout from the matte tap-hole of the furnace is placed in its mouth. The matte is then tapped; and, after filling, the converter is carried back to its stand. In Fig. 2 two spouts are seen to converge to the pit; one for the matte, the other for the slag from the skimming-door. The slag runs into large pots, which are, like the converter, transported by the electric crane from their carriages to the pit, and *vice versa*.

PROF. H. M. HOWE, New York City (communication to the Secretary): There have been few contributions to our knowledge of the chemistry of copper-smelting, since those of Le Play and Plattner, so important as that which Mr. Keller has given us in this admirable paper. A particularly happy as well as reasonable point is his explanation of the greater removal of arsenic in the converter- than in the reverberatory-process; that in the reverberatory-process this element protects itself from elimination by sinking down into the metallic copper or "bottoms," formed in the conversion of white metal into blister, which bottoms are protected from the oxidizing action of the atmosphere by the layer of molten sulphide lying above; while in the converter-process not only the sulphide, but also the molten copper into which the arsenic retreats, is exposed to the action of the blast.

It seems to me unfortunate to speak of the process of Nicholls and James as a direct process. The Welsh process, of which it forms a modification, is carried out with different degrees of complexity in different places. A very common type

with which we may compare that of Nicholls and James is: (1) to carry the copper forward by a succession of calcinings and smeltings to the condition of white metal; (2) to convert that white metal into blister-copper by a single operation, known as "roasting;" (3) to refine the blister-copper. For (2) and (3) Nicholls and James substitute (2*a*) dead-roasting ("calcining") part of the white metal, and (3*a*) melting it with the rest of the white metal, and refining them in the same furnace. But even if (2*a*) be simpler than (2), (3*a*) is surely more complex than (3); and not a few experienced smelters will probably join in my query as to whether (3*a*) should not really be split into two distinct operations.

In saying this I do not mean to criticize the process, which, from the data put before us, appears to have merit; but I regret that it should appear in the position of resting, not solely on its own merits, but on a claim to great directness to which it is not, in my opinion, entitled, especially if we compare it with either the cupola-process, as practiced in America, or the Manhes or converter-process.

Judging from the rate at which the Manhes process has been displacing the Welsh process, it seems to me hardly likely that so slight a modification of the Welsh process as that of Nicholls and James constitutes is likely, in turn, to displace the Manhes process.

The process of Nicholls and James appears to me very much like that which I practiced, and which was then habitual, in Chile, in 1877. There a high-grade matte, approximately white metal, was smelted along with very rich oxidized ores in a regular Welsh reverberatory smelting-furnace, technically known as a "roaster," yielding metallic copper of about 96 per cent., known as Chile bars. The chief difference between this old and widely-known process and that of Messrs. Nicholls and James is, that they use an artificial oxide of copper in place of the natural oxide used by their predecessors.

Mr. Keller's valuable experiments on the removal of impurities in melting given copper oxides and copper sulphides in crucibles seem to me, if I understand his data, hardly to do justice to the Nicholls and James process. He appears to give us the removal which occurs when oxides and sulphides are melted in crucibles. Now, cannot Messrs. Nicholls and James

reply that, in their modification, a great deal of removal of impurities occurs in the calcining by which the oxides are formed, and that Mr. Keller's presentation leaves this removal wholly out of sight?

The comparisons between the purification effected by the converter and that effected by the reverberatory are very important. Grouping together the removals given by Mr. Keller in his final table, and omitting from the comparison, as not fairly comparable, the case of selenium and tellurium of the last two columns, we get the following results:

	Removal in Reverberatory Method.	Removal in Converter.	
		Min.	Max.
Pb	99	95	99
Bi	54	94	96
Sb	50	62	73
As	21	73	91
Se and Te	60	57	71

On the face of this exhibit, the reverberatory stands on a par with the converter as regards the removal of lead, but is at a slight disadvantage as regards antimony, selenium and tellurium, at a great disadvantage as regards bismuth, and at an enormous disadvantage as regards arsenic. But this hardly does the reverberatory justice, for the following reason. The reverberatory process may be, and in Mr. Keller's case was, so conducted that one of its operations, called the "selecting-process," yields two separate products simultaneously: (1) a large quantity of rich copper sulphide or matte, technically called "regule," together with (2) a small quantity of metallic copper, called "bottoms." Into these bottoms much of the impurities concentrate, the overlying regule being correspondingly pure. It is often expedient to treat these two products separately, obtaining from the bottoms a small quantity of impure copper, and from the regule most of the copper, relatively pure. It is this copper that is called "Best Selected." While it is perfectly true that this does not eliminate the impurities, it may still be of great commercial advantage by giving us most of

the copper relatively free from impurities; and for this advantage the figures presented by Mr. Keller give the reverberatory method no credit. For, as I understand, the bottom and the regule were not, in his case, kept separate, but were mixed together.

It is interesting to compare Mr. Keller's results on the selecting-process, *i.e.*, the process of making bottoms and matte jointly, with those obtained by Mr. Allan Gibb,* which I have recalculated, to make them comparable with Mr. Keller's. I give them together in the following table:

Concentration of Impurities in Bottoms in Selecting-Process.

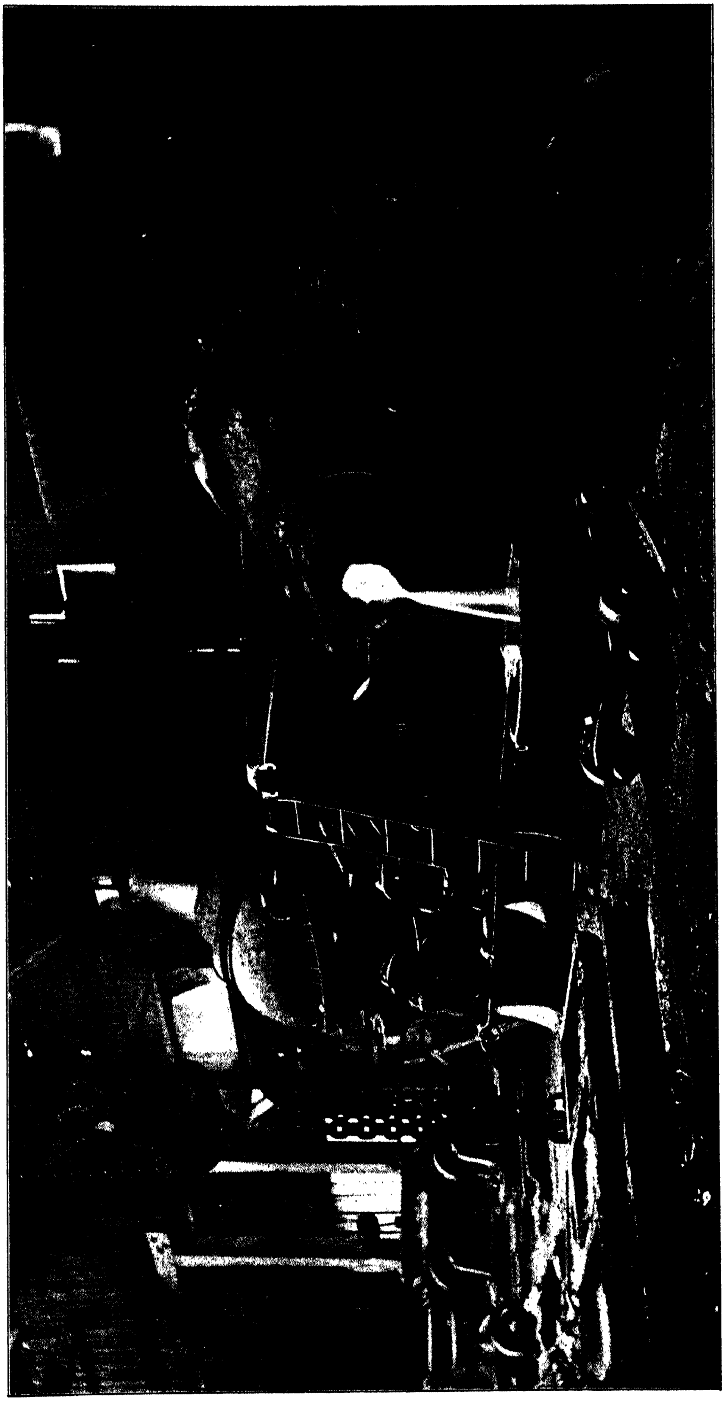
Case No.	Percentage of Total Cu in Bottoms.		Cu.	As.	Sb.	Bi.	Ni.	Pb.	Ag.	Au.	Se. Te.	Authority.
I. ..	8.2	Composition of Bottoms.	97.83	.231	.118	.28	.36		.066	0024		Dean.
		Concentration in Bottoms.	1.	2.62	2.56	1.35	1.		1.926	5.06		
II....	16.0	Composition of Bottoms.	97.63	.571	.432	.063			.061	.0019		
		Concentration in Bottoms.	1.	1.91	4.106	2.61			1.98	6.25		
III....		Composition of Bottoms.		.343	.265	.128		.398			.0022	Keller.
		Concentration in Bottoms	1.	5.70	1.68	1.75		.45			.16	

NOTE.—"Concentration in bottoms" means, for instance, that in Case I., for each 1 per cent. of the initial copper which went into the bottoms, 2.6 per cent. of the original arsenic also went into them.

We notice here that the concentration of arsenic in the bottoms is much greater in Mr. Keller's case, and that of antimony much less than in those of Mr. Dean. Thus Mr. Dean finds that only from 1.91 to 2.62 per cent. of the initial arsenic are carried into the bottoms for each per cent. of the original copper which went into them, while Mr. Keller finds that 5.70 per cent. of the original arsenic is thus carried into the bottoms for each per cent. of the original copper which went into them. Again, Mr. Dean finds that in one case 2.56, and in the other, 4.106 per cent. of the initial antimony is carried into the bottoms for each per cent. of the initial copper carried into them, while Mr. Keller finds that only 1.68 per cent. of the original antimony is thus carried into the bottoms for each per cent. of the initial copper carried into them. As there is nothing in the composition of the bottoms to explain this, we naturally in-

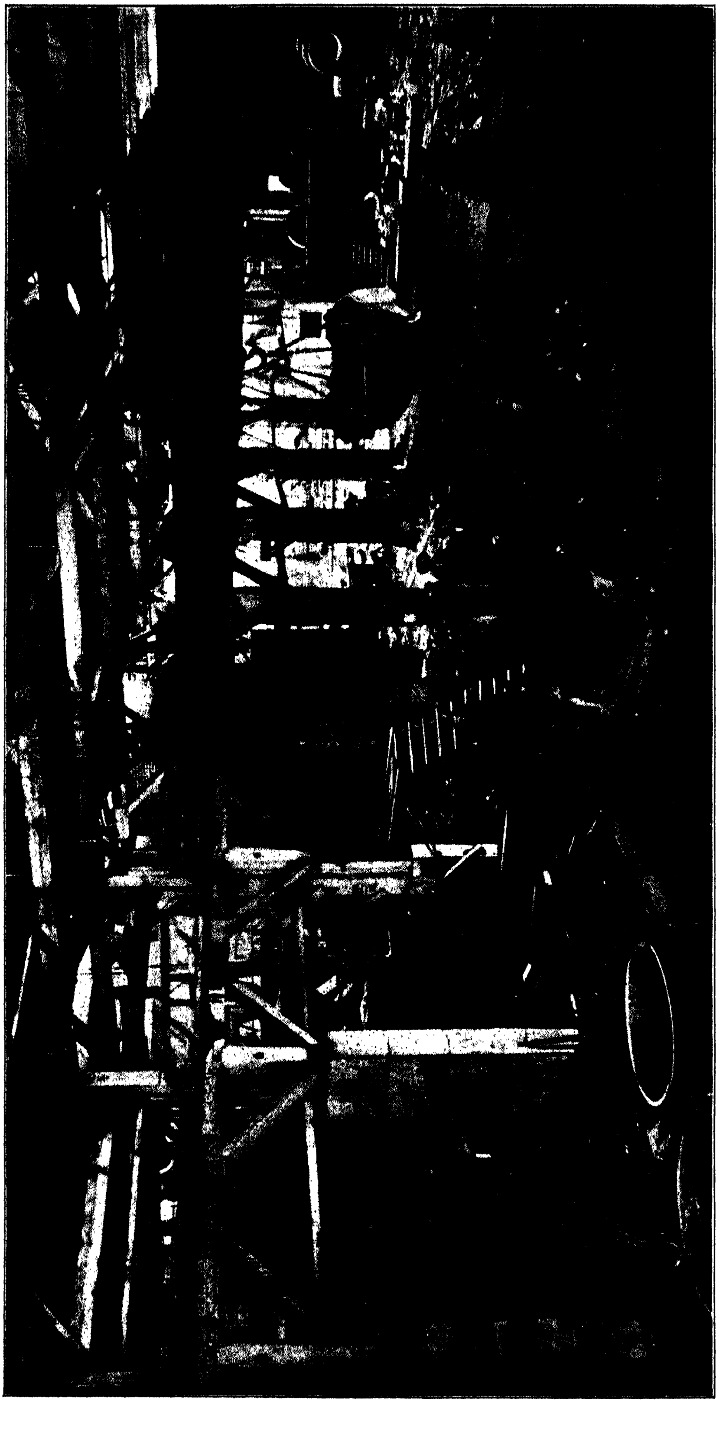
* *Third Report of the Alloys Research Committee of the Institution of Mechanical Engineers*, April 26, 1895, p. 267.

Fig. 1.



Converters in Operation on Copper Matte, at the Works of the Montana Ore Purchasing Co., Butte, Montana.

FIG. 2.



Converter in Transit to be Filled at the Matting-Furnace, at the Works of the Montana Ore Purchasing Co., Butte, Montana.

fer that the degree of concentration of the impurities is subject to great variation. Mr. Keller gives the elimination of gold and silver from the sulphide as 58.46 and 4.65 ounces per ton,* respectively. Mr. Dean found concentrated in the bottoms, in these two cases, 41.5 and 100 per cent. of the gold and 15.8 and 31.7 per cent. of the silver.

MR. KELLER (communication to the Secretary): Professor Howe's remarks on my paper, and the interesting additional information which he gives on the subject, call for comments on but a few points from my side.

With reference to the Nicholls and James process, which Prof. Howe justly considers to be not a "direct process," as it has been styled, he says:

"Mr. Keller's valuable experiments on the removal of impurities in melting given copper oxides and copper sulphides in crucibles seem to me, if I understand his data, hardly to do justice to the Nicholls and James process. He appears to give us the removal which occurs when oxides and sulphides are melted in crucibles. Now, cannot Messrs. Nicholls and James reply that in their modification a great deal of removal of impurities occurs in the calcining by which the oxides are formed, and that Mr. Keller's presentation leaves this removal wholly out of sight?"

This is a correct interpretation of my data; but in reply to the query I would point to other data, on pages 3 and 4 of my original paper (*ante*, pp. 129, 130), where I gave figures, obtained by experiment, showing that in the complete calcination of a matte the removal of antimony is 5 and that of arsenic 15.5 per cent. I took it as a matter of course that the removal of the other elements is less. This assumption was later supported by finding the relative elimination, or removal, of the elements in matte-calcination to be:

	Cu.	Zn.	Pb.	Bi.	Sb.	As.	Se, Te.
Per cent., . .	1	1.86	1.88	1.60	8.15	27.16	2.8

I further pointed out the probable elimination in the actual calcination, where the sulphur is reduced from 23 to 15 per cent., to be:

	Cu.	Zn.	Pb.	Bi.	Sb.	As.	Se, Te.
Per cent., . .	0.18	0.34	0.34	0.29	1.47	5	0.50

In the Nicholls and James process the matte, or white metal,

* NOTE BY THE SECRETARY.—These figures should represent percentages, not ounces per ton. This is printed correctly on page 133 of the present volume.

does not require complete calcination, but it requires a greater degree of it than the one from which my data were obtained. The amount of removal of impurities in Nicholls and James's calcination lies, therefore, somewhere between the two; *i.e.*, for antimony between 1.47 and 5 per cent., and for arsenic between 5 and 15.5 per cent. For the other elements, it is an entirely negligible quantity.

From the above it appears to be clear that the results of elimination by the Nicholls and James fusion are but slightly altered by the introduction of the results of the calcination. The latter plays but a small part in eliminating impurities. It is the fusion of the oxides and sulphides for which special merit is claimed by the originators and advocates of the process as producing extraordinarily pure copper.

With reference to the reverberatory or Welsh process, Prof. Howe compares my results of elimination of that process with those of the converter, and then says:

"But this hardly does the reverberatory justice, for the following reason. The reverberatory process may be, and in Mr. Keller's case was, so conducted that one of its operations, called the 'selecting-process,' yields two separate products simultaneously: (1) a large quantity of rich copper sulphide or matte, technically called 'regule,' together with (2) a small quantity of metallic copper called 'bottoms.' Into these bottoms much of the impurities concentrate, the overlying regule being correspondingly pure. It is often expedient to treat these two products separately, obtaining from the bottoms a small quantity of impure copper, and from the regule most of the copper, relatively pure. It is this copper that is called 'Best Selected.'"

To this I would reply that while it is true that my figures do not do justice to the best-selected copper, the bottoms on the other hand would make a correspondingly worse showing; or, more definitely stated, if the bottoms and sulphides be separately treated, a mere shifting of the impurities from one part of copper into the other occurs, and the sum total of elimination of impurities must be the same as when the two products are treated together, *i.e.*, that my figures hold good for either method.

The statement that we obtain "from the bottoms a small quantity of impure copper, and from the regule most of the copper, relatively pure," is rather an indefinite one. From Mr. Dean's figures, given by Prof. Howe, we are able to gain somewhat more positive information on this point. From his table I take one example (II.):

Concentration of Impurities in Bottoms in Selecting-Process.

Percentage of Total Cu in Bottoms.		Cu.	As.	Sb.	Bi.	Ag.	Au.
16.0	Composition of Bottoms.	97.63	0.571	0.432	0.063	0.061	0.0019
	Concentration in Bottoms.	1.	1.91	4.106	2.61	1.98	6.25

From this it follows that of the total amount of copper and impurities in the matte the following quantities go to the bottoms and into the sulphide:

Distribution of Original Copper and Impurities Contained in Matte.

Percentage	Cu.	As.	Sb.	Bi.	Ag.	Au.
In bottoms................	16.0	30.56	65.70	41.76	31.68	100.0
In sulphide...............	84.0	69.44	34.30	58.24	68.32	0.0

In so apportioning the impurities, the elimination is not yet considered. May we not for this purpose plausibly apply my own figures for the elimination in the total reverberatory process to the impurities in the sulphide, and thus obtain a tolerably fair approximation to the purity of the best selected copper?

The degree of elimination for the elements in question is as follows:

	As.	Sb.	Bi.
Per cent., ...	21	50	54

From the above the composition of the best selected copper for this special example is found:

Amount of Original Contents of Matte Contained in Best Selected Copper.

	Cu.	As.	Sb.	Bi.	Ag.	Au.
Per cent., ...	84	54.86	17.15	26.79	68.32	0.0

Besides these elements, the best selected copper will also contain a large portion of the selenium and tellurium, when these are present in the matte.

In this way the merits of selecting become more definite; but its value for argentiferous material seems small, in this period of electrolytic copper-refining.

It should further be remembered that, with decreasing quantity of impurities in the matte, the degree of concentration in the bottoms diminishes, and that when we have made a large percentage of the latter, the remaining sulphide is still comparatively impure and rich in silver. Of all the elements, gold alone is completely eliminated.

It would be interesting to know if Prof. Howe, with probably all original literature at his disposal, could have shown a comparison, derived from an actual test, between the composition of a matte and the resulting bottoms and best-selected copper. Percy, who wrote when the process was important, gives us no information except the analyses of bottoms. It is also interesting to note what he says of the best-selected copper of his time:

"*Best-selected copper* commands the highest price in the market, and is in special request for the manufacture of certain alloys, such as the best qualities of brass, and the white alloy known as German silver; it is prepared, as we have seen, by a special process of purification; and it is generally pretended that it is exclusively produced from the purest ores. That some smelters do employ the purest ores in making this variety of copper may be true; but that others do not, seems very probable from the fact, which is notorious in Birmingham, that copper which is occasionally (I might say, frequently) sold as best selected, is extremely bad."*

The selecting-process, in connection with converting, has recently been recommended by Mr. Paul David, of France, and for it he has constructed a modified converter; calling it a "sélecteur."†

What can be accomplished by the converter in this direction may well be illustrated by the example of an under-blown converter-charge. We have the following

Composition of Matte, and the Resulting Copper or Bottoms and Regulus in an Under-Blown Converter-Charge.

	Matte. Per Cent.	Regulus Per Cent.	Copper or Bottoms. Per Cent.
Cu	49.34	81.40	99. +‡
Pb	0.0738	0.0071	0.0087
Bi	0.0337	0.0012	0.0056
Sb	0.1010	0.0365	0.0992
As	0.0480	0.0081	0.0260
Se, Te	0.0021	0.0084	0.0020
	Ozs. per Ton.	Ozs. per Ton.	Ozs. per Ton.
Ag	14.60	15.60	43.00

* Percy's *Metallurgy*, 1861, p. 364. † E. and M. J., lxvi., Oct. 22, 1898.
‡ Fraction unknown.

Of these products the weights are unknown, but from the knowledge of their copper- and silver-contents the concentration and the proportion of copper and impurities in bottoms and regulus may be calculated, it being assumed that the losses of the two metals are of negligible magnitude. In the way thus indicated the following results are found:

Degree of Concentration.

	Cu.	Pb.	Bi.	Sb.	As.	Se, Te.*	Ag.
In copper or bottoms........	1	0.06	0.08	0.48	0.27	0.47 (?)	1.46
In sulphide or regulus......	1	0.06	0.02	0.22	0.10	2.39 (?)	0.64

Distribution of Original Copper and Impurities.

Percentage	Cu.	Pb.	Bi.	Sb.	As.	Se, Te.	Ag.
In copper or bottoms.....	44.	2.64	3.52	21.12	11.88	20.68 (?)	64.20
In sulphide or regulus...	56.	3.36	1.12	12.32	5.60	133.84 (?)	35.80
Eliminated.................	0.0	94.	95.36	66.56	82.52	(?)	0.0

It will at once be seen that bottoms from the converter differ widely from the bottoms of the reverberatory; inasmuch as there is no positive concentration in the former excepting that of the precious metals, while in the latter arsenic, bismuth and antimony are also considerably concentrated. In the converter the impurities in the bottoms, as well as those in the regulus, show a high degree of elimination.

The regulus, in being blown to metallic copper, would naturally be freed of another portion of its impurities, but to no greater extent than if the bottoms remained with it. The bottoms, on the other hand, being prematurely withdrawn, will carry a greater portion of the impurities than the same amount of copper would do if left in the converter to the normal finish, since the blast acts on it as well as on the sulphide. From this it follows that by "selecting" in the converter, the sum total of elimination of impurities is smaller than in the usual nor-

* It is a well-established fact that selenium and tellurium concentrate in the sulphide. In this case, however, the apparent concentration is aggravated by an analytical error in determining one or the other of the products.

mal blowing, and that for argentiferous copper, which is to be electrolyzed, nothing is gained by such a practice. When normally-blown converter-copper is too impure for electrolysis, it needs to be diluted with a lower-grade copper. The quantity of the latter necessary in this case is not greater—I believe it is less—than the quantity necessary to dilute the bottoms of the selecting-process to the necessary grade. There is another reason, however, for my opinion that selecting is a step in the wrong direction. It is based on the results, shown in my original paper, of the effects of over-blowing the charge in a converter, and thus attaining a very high degree of elimination of the impurities not attainable in a reverberatory furnace.

With regard to the fact, pointed out by Professor Howe, that Mr. Dean's figures on the concentration of arsenic and antimony in the bottoms do not harmonize with mine, or even amongst themselves, I beg to say that a rational explanation may be found in the composition of the original matte. In such mattes as I have tested the quantities of the two elements do not differ very widely, and I have uniformly found a greater degree of concentration for arsenic than for antimony. Now if, in Mr. Dean's mattes, the latter element greatly preponderated, a change, as shown by his figures, might be anticipated. Only when two elements are present in approximately equal quantities may their properties be compared; otherwise, we are liable to find rules reversed.

Prof. Howe (communication to the Secretary): Replying to Mr. Keller's question, I do not now remember any case in which the composition, both of the charge and of the resultant bottoms and regule, is given. The nearest to this, so far as I know, is on pp. 304 and 305 of Le Play's *Description des Procédés Métallurgiques Employées dans le Pays de Galles pour la Fabrication du Cuivre.* He gives the composition of the slag, bottoms and regule; but I do not find that he gives the composition of the charge itself. Part of his results are condensed on page 66 of my "Copper Smelting" (*Bulletin No.* 26, *U. S. Geol. Survey*).

The Manganese Ore-Industry of the Caucasus.

Postscript to the Paper of Frank Drake, New York City. (See p. 191.)

(Buffalo Meeting, October, 1898.)

Since the presentation of this paper at the Atlantic City meeting, I have received statistics of the exports of Caucasian manganese-ore which I had not been able to procure before, and which slightly modify, as well as complete, the statements made in the paper on that subject.

The following table, showing the exports from Poti and Batum during the last five years, is taken from the British consular reports:

Exports of Caucasian Ore (Long Tons).

To.	1893.	1894.	1895.	1896.	1897.
Great Britain	42,930	65,110	60,616	77,754	68,650
France	4,100	....	150	5,650	....
Russia	....	9,890	9,600	20,175	28,446
Belgium	3,125	2,520	....	220	....
Germany	40,405	51,455	59,565	58,825	70,810
United States	36,070	28,300	55,787	3,600	42,200
Total exports	126,630	157,275	185,718	166,224	210,106

It will be seen that Great Britain, though still a large importer, has yielded the first place to Germany.

An Apparatus for the Removal of Sand from Waste-Water of Ore-Washers.

Discussion of the Paper of J. E. Johnson, Longdale, Va. (See p. 225.)

R. W. Raymond, New York City: From a study of Mr. Johnson's paper, I conclude that his apparatus is simple in construction and operation, and imitates in a revolving machine the movements of hand-shoveling. It does what a man would do in removing sand from the waste-water with a perforated shovel. I am sorry to say that I fear the present condition of the brown-ore mining industry in the East, as long as

the Mesabi and other outrageous deposits of the West continue to be productive, is not such as to need very much machinery. I do not at this moment call to mind any places where brown ore is washed on a large scale, outside of Longdale, though there may be such.

There is no question, however, of the advantage of removing heavy sand from the waste-water, so as to leave nothing but the clay. And this advantage is not confined to the washing of iron-ore. I think Mr. Johnson's device might be applicable in other parts of the country and to other purposes. We have a problem, for instance, in the West, in the handling of tailings from stamp-mills and from hydraulic mining. The mass of the tailings consists largely of heavy sand, then of clay, which will remain suspended a long time in water, then of water. Now, when we have got to impound those tailings, as we have to do in many places, we shall fill up our reservoirs with very much greater speed if we have to run all the sand into them. The amount of sand in our reservoirs of tailings in the West is so large that little else can be seen. If we are obliged to run that where we run the water, and hold it with an expensive dam, the rate at which we fill up our available reservoir space is a very serious matter economically. If we could shovel up the sand by some such device as this, and put it where it would not be in danger of being carried down any stream to bring about *débris* damages and litigation, then we should have, remaining in our waste-water, a suspended clay which would fill up the reservoir very much less rapidly, and do very much less damage, if the dam did not remain perfectly tight.

I was also interested to note in Mr. Johnson's paper the practical reasons why other devices which would naturally suggest themselves did not work so well when they came to be actually tried. One would say that the getting of sand out of water was so very simple; one might run it into a tank, and let it settle, and draw off the supernatant water. But none of those obvious things has seemed to be as effective or as simple or as cheap as this.

JOHN BIRKINBINE, Philadelphia, Pa.: It has been my privilege to observe the conditions at the Longdale works, and to see why Mr. Johnson had to find some means of handling the sand. I cannot agree exactly with Dr. Raymond's first

proposition, but the second I do agree with. Ores are being washed in considerable quantity elsewhere than at Longdale. Large quantities are being washed in Alabama, Tennessee, and some in our own State of Pennsylvania. The second proposition I thoroughly agree with. The question that arose in my mind was whether the same general plan of apparatus could not be applied to the material which flows from the coal-washers in the anthracite region. I assume that the specific gravity of the removable material will be less; but if a considerable portion of the coal-waste which is giving so much trouble to farmers and to the cities along the rivers could be retained at the works in an economical way by Mr. Johnson's apparatus, or a modification of it, a great benefit would result.

E. D. PETERS, JR., Dorchester, Mass.: It has seemed to me, in listening to Mr. Johnson's paper, that that apparatus might have considerable value in cyaniding in the West. The great difficulty in cyaniding, as every one knows, is the fact that the slimes come out at the same time with the sand, and that, when they settle in the tailing-beds, the slimes form a layer which in a little while becomes like gutta-percha. Consequently, many operators are settling their coarser sands in tanks. In Arizona and the Southwest, where wages are $3 and $3.50 a day, this involves a very serious expense, which they are trying to avoid by separating the coarser sands with pointed boxes. But that means a considerable loss of water, which runs out with the sand all the time, and which, in those regions, is an important matter; and I do not see why some such apparatus as that described by Mr. Johnson would not be a valuable addition to at least a dozen mills in the southwestern part of the country that I could name to-day that are cyaniding their tailings.

DR. RAYMOND: I would like to ask Dr. Peters whether he does not think that in that case there would be a difficulty in using that apparatus. Would not the sand lying on the shovels of the apparatus act as a filter to hold back the slimes?

DR. PETERS: The great portion of the slimes would, I think, remain in the water. We could stand 5 or 6 per cent. of slimes without any trouble.

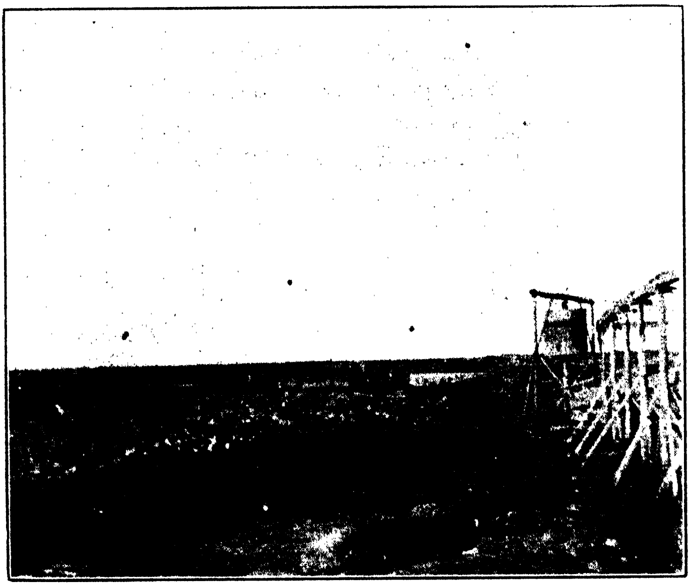
The Kotchkar Gold-Mines, Ural Mountains, Russia.

Discussion of the Paper of H. B. C. Nitze, Baltimore, Md., and C. W. Purington, Boston, Mass. (See p. 24.)

(Atlantic City Meeting, February, 1898.)

PROF. HENRY LOUIS, Newcastle-on-Tyne, England (communication to the Secretary): I have read this paper with much pleasure. It presents a very accurate summary of a very interesting district. Like the authors, I was struck by the extreme

FIG. 1.

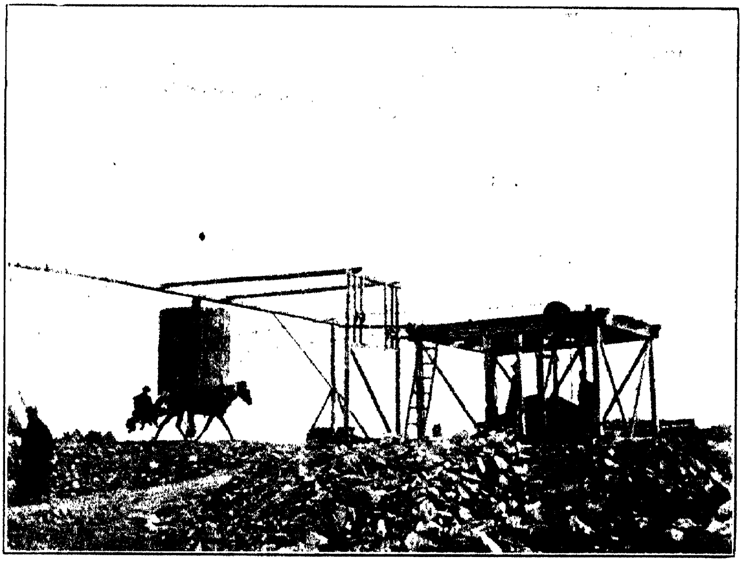


General View, near French Co.'s Cyanide Works, Showing Horse-Whims, etc.

flatness of this region, which looks strange to a gold-miner accustomed to other reef-districts. This prevailing absence of relief in the contours is shown in Figs. 1 to 4, from photographs taken in August, 1897, some of which also show the horse-whims used here for hoisting. The flatness of contour is due, no doubt, to the fact that the rocks are similar in structure over large areas and are easily decomposed. I am quite at one with the authors in looking upon the so-called "placers" as the result of decom-

position *in situ* of the gold-bearing veins and rocks; in a sense, these "placers" might almost be described as the gossans of the gold-bearing deposits. The products of decomposition are prevented from traveling far by the flatness of the land, and the same reason accounts for the fact that the very fine gold thus liberated still remains *in situ*. Gold-miners will readily understand how fine the gold must be, and how different from typical shallow-placer gold, when the fact is mentioned that the unconcentrated tailings from the sluice-boxes paid handsomely to chlorinate and to cyanide.

FIG. 2.



Horse-Whim at Kotchkar.

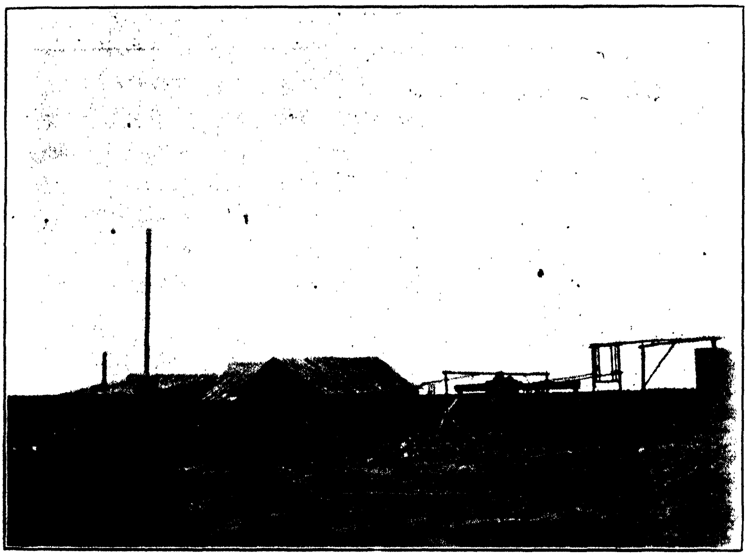
I agree pretty nearly with the views expressed by the authors as to the genesis of the primary deposits. These seem to be narrow veins, deposited in the fissures of the shattered granite, which is in places (*e.g.*, in the Ouspenski mine) silicified for a width of 2 to 3 feet on either side of the quartz vein. The granite is mineralized to the same width, and is taken out and milled. The quartz veins here pass gradually, without any clear line of demarcation, into the silicified granite.

In the French Company's mine the quartz veins seem to be rather harder, and the walls are better marked, the granite forming the walls being softer and less silicified. In this mine,

I noticed a strong N.–S. cross-vein, which seemed also to carry gold, cutting through the main vein-system at right angles, and heaving the latter some yards. It is therefore evident that faulting does occur.

As the gold-contents of the country-rock extend to a limited distance only on either side of the vein, I hold that fissures in the granite formed the channel through which siliceous and auriferous solutions traveled, altering more or less the country-rock on either side, and forming the auriferous quartz-

FIG. 3.



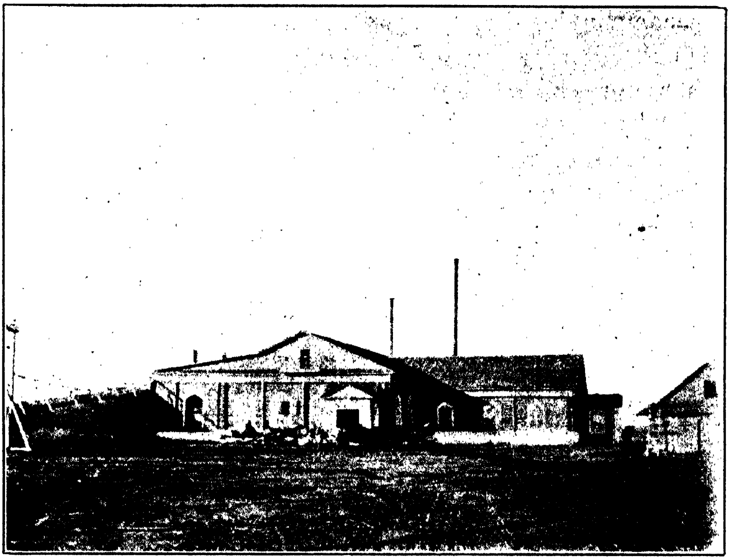
Shaft-House and Mill at Zelinkoff's Ouspenski Mine.

veins proper. Whence the gold came originally, whether from the surrounding mass of the granite or from some remote source, there is no evidence to show.

The development of the Chilian mill for gold-milling in this district is probably unique, and the results, in working on oxidized ores, are by no means unsatisfactory. A mill crushing on an average 18 tons per 24 hours requires 6 to 7 H. P. to drive it. It is true that the ore is rather soft, about three-fifths of the total amount consisting of altered granite. As suggested by the authors, the extended use of the Chilian mills is probably due to the very inferior character of the only stamps hitherto available in this portion of Siberia. The largest, if not the only, stamp-mill in all this region is at

Beresov. Here there is a 30-stamp mill with stamps weighing 600 to 700 pounds, and running at 65 drops per minute. There is no true battery-box, the coffer being a primitive form of the old stamp-coffer with cast-iron bottom and wooden sides. Of course, there is neither rock-breaker nor automatic feeding. The total crushing capacity of the mill is about 50 tons per 24 hours; and the number of men and lads employed is 17 per shift, exclusive of foremen. No mercury is used in the boxes; the pulp is run over blanket-strakes, 25 feet long and 1 foot wide, followed by similar amalgamated-copper strakes, arranged

FIG. 4.



French Company's Large Mill.

like blankets, so that one plate is always dry while its neighbor takes the entire current of pulp. It is hardly to be wondered at that they find their plates to catch but little gold. The blanketings are buddled in small box-buddles, and the concentrates thus obtained are rubbed up with mercury in the same buddle. The amalgam thus produced is finally washed clean. It is asserted that 95 per cent. of the total gold production of this mill is obtained in this way from the blanketings. Obviously, this method of milling is primitive in the extreme; and it is not surprising that, in comparison with such a type of stamp-mill, the well-arranged Chilian mills described by the authors receive the preference.

The Kytchtym Medal.

Discussion of the Papei of Dr. Persifor Frazer, Philadelphia, Pa. (See p. 613.)

(Buffalo Meeting, October, 1898.)

O. S. GARRETSON, Buffalo, N. Y.: If I may judge from the half-tone illustration engraved from a photograph of this medal and accompanying Dr. Frazer's paper, I do not think the casting is exceptionally fine. I have often made castings of similar character which I think were equally good. For instance, I have taken a coin, moulded it directly in fine sand, and cast from it in Scotch (or the equally good, if not better, "American Scotch") foundry-iron a copy quite as sharp and smooth as the original. The secret is in using very fine sifted sand. In fact, I have used fôr this purpose the dust which had accumulated on the beams overhead in my foundry, riddling it with a very fine sieve before using it.

[NOTE BY THE SECRETARY.—After the session at which these remarks were made by him, Mr. Garretson cast at his Buffalo foundry, in the presence of a number of members of the Institute, copies of the bronze pin furnished to visiting members by the Local Committee of the Buffalo meeting. The accompanying illustration, Fig. 1, has been reproduced from a photograph of such a casting, made in green sand, under the conditions of ordinary foundry-practice, and not subsequently polished in any way. I can also certify that the photographic negative was not touched, to improve its clearness of definition, before the production of the "half-tone" plate from which Fig. 1 has been printed.

As, in the case of the Kytchtym medal, it was the thin, ribbon-like sprue, rather than the low-relief features of the medal itself, which indicated the fluidity of the metal and the sharpness of the casting, so in this work of Mr. Garretson it is the long and slender pin which offers the strongest evidence of these qualities. It is scarcely necessary to point out that the filling in the mould of a long and narrow opening, such as this pin presents, is the severest test that could be imposed upon the casting, and that the sharpness of the point of the pin in the casting is striking evidence of the success with which this test has been met.

The Pittsburgh Testing Laboratory has made for the *American Manufacturer* measurements of this casting and an analysis of the sprue.

The thickness of the main casting is from 0.049 to 0.068 inch. That of the tag bearing the stamped number is 0.024 inch. The analysis, which represents the iron ordinarily employed by Mr. Garretson for thin castings, is as follows:

	Per cent.
Combined carbon,	0.63
Graphitic "	3.02
Manganese,	0.27
Phosphorus,	0.482
Sulphur,	0.079
Silicon,	1.95

R. W. Raymond, New York City: The analyses of the iron of this medal, communicated in Dr. Frazer's paper, certainly do not indicate any peculiar composition likely to confer upon the iron extraordinary fluidity or freedom from shrinkage in the mould. Nor is the medal itself (which I had, with others, the opportunity to inspect at the Atlantic City meeting) comparable for delicacy with the famous castings from Ilsenburg, in the Harz. The latter I have always supposed to be made from iron of special composition; and, having seen them only in their final form, as they appear in the market, I cannot positively say that they receive no final polishing, though they have no traces of such treatment, and I confess that it seems to me almost impossible that it could have been applied without leaving some recognizable sign. The artistic work of Ilsenburg was illustrated in a magnificent separate exhibit at the Columbian Exposition of 1893, in Chicago. It was superior in fineness of detail, though not in vigor and originality of design, to the Russian cast-iron statuettes which attracted so much attention at the same Exposition.

Some years ago, the Hecla Iron Works, Messrs. Poulsen and Eger, of Brooklyn, N. Y., exhibited artistic castings in iron which closely approached the Ilsenburg standard. I remember that thin patterns were shaped and first cast in plaster by artists of special skill, and that the firm maintained, for the education of its employees, a regular school of design.

The Hecla Iron Works are still in existence and continue to furnish, as they have done for years past, the architectural and ornamental iron-work of many New York interiors, but, not having heard lately of any small and delicate castings for other purposes produced by this concern, I have made direct inquiry, which has elicited the following interesting letter from Mr. N. Poulsen:

Dear Sir: In answer to your inquiry as to whether we are still making artistic castings such as we made some fifteen years ago when you visited at our place

of business, I wish to say that when we started our business (some twenty-odd years ago) we immediately aimed at making our castings for architectural work the best that could be produced by skilled workmen. Some time before you called on us I had made a trip to Europe, had there seen some very beautiful work of this kind, and, on my return, had brought with me samples purchased in Paris, Berlin and other European cities, but upon further inquiry found that they all came from Ilsenburg in Germany. I wanted to see if we could not make as good castings in this country, and went about it in this way:

As neither Mr. Eger nor myself is a chemist, we were unable to analyze the iron, and therefore employed a mining engineer, Mr. Charles W. Miller (a member of your Institute), who had just graduated from Columbia College. We arranged for him a little chemical laboratory in our place of business, for the purpose of analyzing the quality of the iron in those German castings, and the different kinds of iron found in this country adaptable for foundry-purposes. He found that the iron in the German castings was not superior to that we had in general use in our foundry. We then, together with him, studied publications on foundry-work in regard to facings, sand, etc., and had him superintend experiments in our foundry. The foreman of the foundry did not like the idea of a young man fresh from college coming to show him how to make better castings, and repudiated the whole idea. But we went ahead; selected fine patterns from our stock; had facing and sand prepared according to the best information we could get; put the patterns into the hands of some of our best moulders, and established Mr. Miller in the foundry to see that everything was carried out as he thought best. The result was a complete success. The castings we turned out were, I think, handsomer than those I had brought with me from Europe. We established the fact that we had the proper material, and also the skillful pattern-makers and moulders required.

The moulders got very much interested in their own work after seeing the fine results—so much so, that they many times, together with ourselves, waited in the foundry until late in the evening, when the castings got cool enough to take out of the sand, just for the pleasure of enjoying their own handsome work.

These experiments were the means of improving our ornamental castings for architectural work, which was what we were aiming at. Our work was generally recognized by architects to be far superior to that of our competitors at that time; and for many years after there was no other American establishment in our line of business who could produce such work. Now there are in this country many concerns which turn out very handsome castings, but I think we may fairly claim that they are all, more or less, the outgrowth of ours—employees of ours having either started out for themselves or taken positions in establishments already existing. Our little experiments have thus borne very good results. I think that to-day the ornamental iron-work for building-purposes produced in this country is superior to that from any other country, and is so recognized abroad.

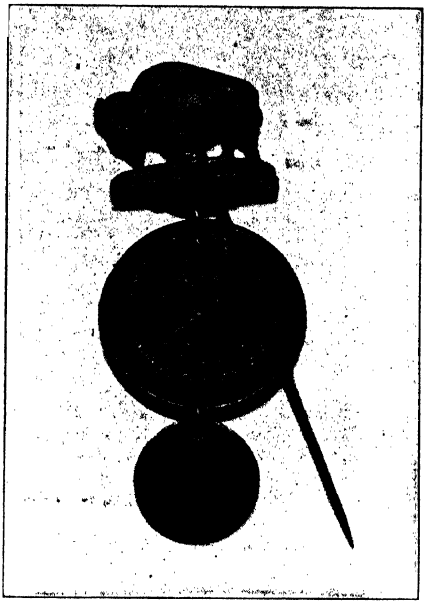
The school of design which we formerly maintained for our employees has been given up as being no longer necessary, in view of the greatly-increased opportunities for such instruction now open to ambitious mechanics in trade-schools, etc. At that time there was only the Cooper Union in New York, and nothing on Long Island.

The fine artistic castings made by us in those days are still useful in our work, though we no longer produce them for the market. When, for any reason, we are dissatisfied with the work done by our pattern-makers and moulders, we bring out some of these old pieces, and tell them that what the foundry has done once it ought to be able to do again.

Yours truly,

N. Poulsen.

Fig. 1.

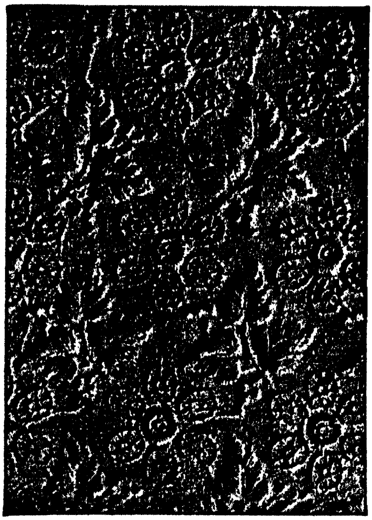


Casting made by Mr. Garretson of the Pin of the Buffalo Meeting.

Fig. 2.

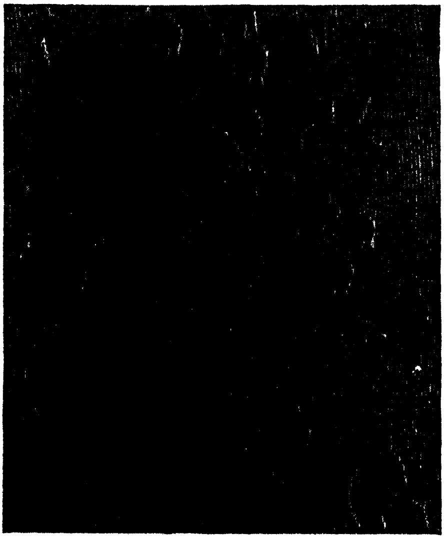
Casting in Hard Iron upon Lace Laid on Sand.

FIG. 3.



Casting in Gray Iron upon a piece of Thin Summer Dress Goods with Machine Embroidery.

FIG. 4.

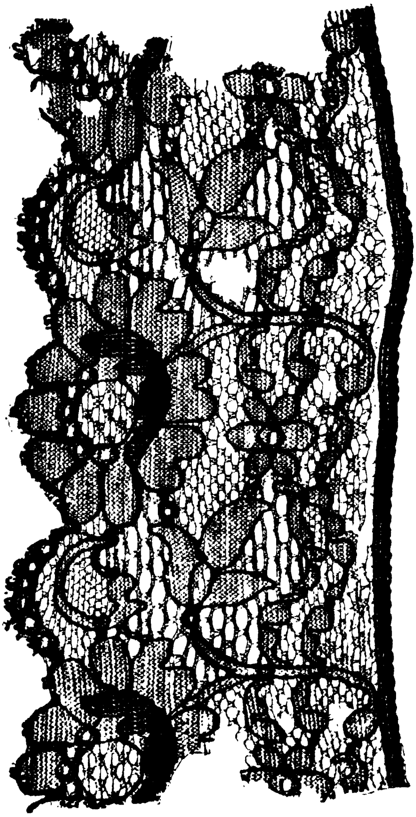


Top of Cardboard Box Embossed upon a Cast-Iron Die, made by pouring Molten Metal on Embroidered Piquet.

I am sincerely glad of the opportunity to put on record this example of American ingenuity and enterprise, and to give the credit due to the pioneers of a great improvement in American foundry-practice.

In this connection, a very interesting illustration of delicate

FIG. 5.



Piece of Carbonized Lace upon which Molten Iron has been Poured and a Perfect Replica Obtained Therefrom.

The fine threads are not thicker than cobweb, and the same lace can be used again, though slightly imperfect.

castings in low relief was furnished in 1887 by the remarkable reproductions in cast-iron of carbonized lace, etc., for which Mr. A. E. Outerbridge, Jr., of Philadelphia, received from the Franklin Institute the John Scott Legacy Medal and Premium.*

* See *Jour. Franklin Institute*, vol. cxxiii., page 450, and vol. cxxiv., page 389. Also *The Foundry* for April, 1896.

The novelty of Mr. Outerbridge's work consisted in the method by which lace, leaves and other organic fabrics or structures were previously carbonized, before introduction into the moulding-flask. The objects were placed in a cast-iron box, the bottom of which was covered with a layer of powdered carbon; then another layer of carbon-dust was sprinkled over them; the box, covered with a close-fitting lid, was heated gradually in an oven, to expel moisture, and the temperature was slowly raised until the escape of blue smoke from under the lid had ceased. The box was then heated white-hot, kept in this condition for two hours, and then removed from the fire and allowed to cool. The fabrics, etc., thus carbonized were not brittle, and could be made white-hot before consuming. In making castings from them they were laid smoothly upon a face of green sand in the mould, and the molten metal was poured upon them. In one case a piece of lace was suspended vertically in the mould, and the molten iron was introduced on both sides of it, so as to rise to a common level. When the casting was cold it was thrown upon the floor of the foundry, and separated into two parts, while the lace fell out uninjured, and the pattern was found to be reproduced upon both faces of the casting. For the interesting particulars of this and other experiments I refer to the authorities already cited. The special subject of these remarks is not the remarkable behavior of the carbonized fabrics, but the delicacy with which their details were reproduced, with all the sharpness of electrotypes, in the cast-iron. Figs. 2, 3, 4 and 5, all of which represent the rough castings, will illustrate this point.

The Effect of Sizing on the Removal of Sulphur from Coal by Washing.

Discussion of the Paper of Charles C. Upham, New York City. (See p. 486.)

(Buffalo Meeting, October, 1898.)

Prof. Courtenay De Kalb, Kingston, Ont. (communication to the Secretary): It may be due to some lack of carefulness in expression that Mr. Upham seems to imply that there

is for each coal what he calls a "critical size," that is, some maximum size of crushed particles "at which an almost complete division of the coal and pyrites takes place," advantage of which may be taken by screening as a preliminary to washing, whereby grades may be produced leaving the coal and pyrites particles in such relations to each other as regards their relative velocities of fall in water that a very perfect separation may be attained. There are many coals to which this will apply; but the range is far more limited than Mr. Upham appears to assume. Pyrites occurs in bituminous coals to a comparatively small extent in a form analogous to that in which it occurs in rocks and ores, that is to say as one of the components of a practically homogeneous mineral aggregate, disseminated irregularly through the mass. When pyrites does exist in this state in coal it is usually disseminated in grains of such minute dimensions that crushing to the "critical size" would mean a reduction to so small a diameter for the maximum-sized particles that the majority of the coal would be in a nearly pulverulent condition. Furthermore, it is generally true that, when pyrites is so disseminated, the "essential ash" of the coal will also be high; and as this is irremovable, the case of such coals, so far as betterment by washing is concerned, is hopeless.

Pyrites in general follows the slate partings, that is, the aluminous "accidental ash." For the most part, when well-developed masses of pyrites are met with in coal, they will be found to surround or be attached to slate. This is the case with a large percentage of the "sulphur-balls" encountered in mining our Appalachian bituminous seams. The association of such "sulphur-balls" with "rolls" and "horse-backs" is a conspicuous phenomenon, the geologico-chemical relation underlying which is obvious. I do not mean to assert that "sulphur-balls" will not be formed independently of such distortions of the vein, but merely that when they do occur the pyrites masses are almost invariably present.

Another form in which pyrites very commonly exists in coal-veins is that of films or thin laminæ between the parting-planes of the coal, both vertical and horizontal, but most often on the vertical planes. This so-called "scale-pyrites" is generally so closely adherent to the coal that it will not be wholly

broken loose either in handling during shipment or in crushing. But the portion which does separate comes in the shape of very minute scales; and, as the surface-tension of such scales is very high, they will float. I think Mr. Upham may find that the "float-pyrites" which he mentions was mainly of this sort. This "scale-pyrites" is a marked feature of the coal from the Pittsburgh seam.

My experience does not coincide with that of the author, that the pyrites usually crushes more finely than the coal. My own tests have shown that coals of nearly similar specific gravity and hardness, crushed under like conditions, will produce approximately the same relative amounts of the different degrees of fineness from the maximum size to dust, the pyrites (except when large masses are present) occurring through these several grades in accordance, apparently, with their original size in the coal. The proof of this is found in screening the coal to rather close sizes; determining the percentage of sulphur in these; re-crushing the larger sizes separately, and then re-screening the crushed products, and determining their sulphur-contents. I have yet to find a case in which the sulphur-content will not be greater in the coarser sizes of these re-crushed samples.

The tests which Mr. Upham has instituted are of great importance and practical value; but it seems evident that general principles will be difficult to deduce. Each coal will require special investigation; and such studies, carefully carried out, will lead undoubtedly to a very material improvement in the product of our coal-washeries.

The Electrolytic Assay as Applied to Refined Copper.

Additional Discussion of the Paper of Mr. George L. Heath, Presented at the Lake Superior Meeting, July, 1897. (See *Trans.*, xxvii., 390, 962.)

(Buffalo Meeting, October, 1898.)

Edgar Hall, Tenterfield, New South Wales (communication to the Secretary): Mr. Klepetko* asks for information showing at what percentage antimony and arsenic, as impurities, begin to affect injuriously the quality of brass.

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* *Trans.*, xxvii., 967.

This point was determined in the best English works many years ago, when it was found that brass for tubes or wire or drawn-work could not be made from copper containing more than 0.05 per cent. of arsenic and a trace of antimony (less than 0.01 per cent.). The presence of small quantities of bismuth is quite as injurious as that of antimony, and in unrefined copper is even worse, since bismuth is not eliminated to so great an extent in refining.

The question whether a refined copper is fit to make first-class brass can be determined by the "yellow metal test" more quickly than by analysis.

This test shows by the appearance of the fracture whether antimony is present in sufficient quantity to spoil the brass, less than 0.01 per cent. of antimony causing bright facets to appear on the fractured faces of the yellow metal. The test is performed in the following manner:

The copper to be tested is melted rapidly in a plumbago crucible, and to it is added two-thirds of its weight of high-grade spelter. The metal should be hot enough to pour well after the addition of the spelter, which should be done quickly. The alloy is stirred or well shaken, and poured into a dry, hot mould, 4 inches square by 1¼ inches deep. This mould should *not* be greased or oiled before casting the ingot. The ingot of yellow metal is allowed to cool slowly until it can just be handled, when it may be finally quenched with water. A V-shaped nick is then cut across the middle of the surface, and the ingot is broken in two under a steam-hammer or by any other suitable means.

If the copper is good, the fracture will have a regular, frosted, granular appearance, with no bright facets. The presence of antimony shows itself in bright facets, perpendicular to the base of the ingot, which increase rapidly in number and brightness with increasing quantities of antimony.

In the determination of arsenic and antimony we used to employ Abel and Field's method, somewhat modified. Possibly this has been abandoned of late years for a more rapid method. The arsenic was always weighed as ammonium-magnesium arsenate, and was dried at 100° C., and weighed as such on the filter-paper, which had been previously dried and tared. A correction for solubility was made by measuring the

volume of the filtered solution (without the wash-water) and adding 0.0001 gramme to the weight of the precipitate for each cubic centimeter of solution.

Discussion on Tuyeres in the Iron Blast-Furnace.*

(Buffalo Meeting, October, 1898.)

R. W. RAYMOND, New York City: In connection with the subject of multiple tuyeres, my attention has been drawn to the practicability of gaining, without the multiplication of tuyeres, the advantages which that measure is intended to secure; and learning that this had been attempted in the invention of the Gaines radial-discharge or fan-shaped tuyere, I wrote to Messrs. Gustafson Brothers, of Sequachee, Tenn., who manufacture that tuyere, for information concerning the invention itself and its performance in practice. To their courtesy I am indebted for most of the facts given below. I present these statements without expressing any personal opinion on the subject (having no personal knowledge of it), for the purpose of bringing this device to the notice of members, and calling forth both criticism and testimony concerning it.

Admitting that definite advantages are to be secured by the multiplication of tuyeres, we must still confess that this change involves considerable expense in the increased number of blast-pipes, tuyeres, coolers, connections, and pipes for water-circulation. There is, moreover, a serious question involved in the weakening of the furnace-wall by extra perforations. This consideration applies with special force to furnaces already built without reference to such an increase of tuyere-openings.

Ledebur, in his *Handbuch der Eisenhüttenkunde*, says on this point (I translate from the edition of 1893, page 365):

"A limit to increase in the number of tuyeres is given, in any case, by the circumstance that the perforation of the wall weakens it; in other words, that there must be left between the tuyeres a mass of masonry sufficient to carry the parts above. In most cases an appropriate number may be secured by giving to the furnace as many openings as there are meters in its circumference at the tuyere-level. For instance, for a hearth-diameter of 2.5 meters, *i.e.*, a circumference of 7.7 meters, seven or eight openings might be made, of which, however, one would

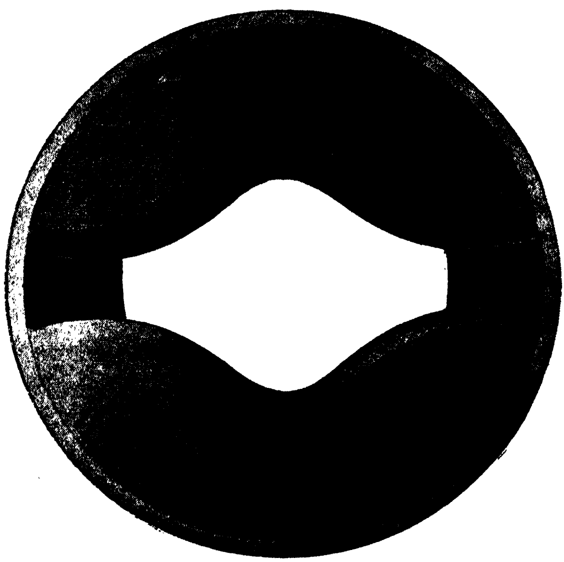
* To this discussion belong the papers of Messrs. Fackenthal and Hartman, pp. 370 and 666 in this volume.—R. W. R.

be occupied, not by a tuyere, but by the fore-hearth or the closed front with cinder-notch."

Obviously, any device which would produce the same effect as the multiplication of tuyeres without requiring change in existing equipment or additional perforation of walls would be highly acceptable, especially to the managers of furnaces already built. This is what is claimed for the Gaines tuyere.

Figures 1, 2 and 3, giving front, back and side views of this tuyere, explain themselves. Its essential feature is the horizontally elongated discharge-opening, which spreads the blast

FIG. 1.

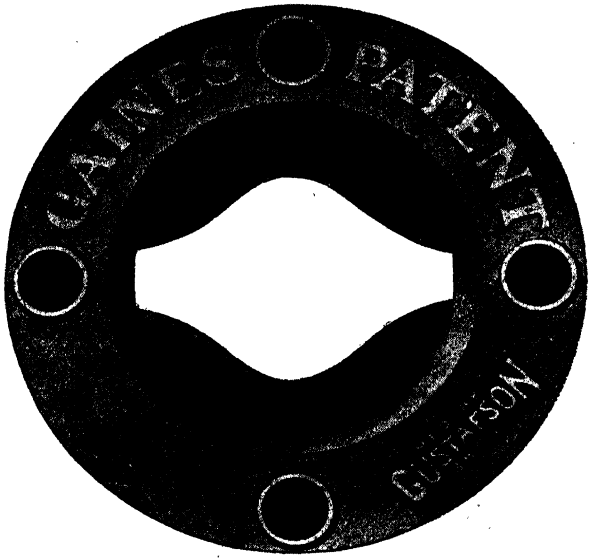


Gaines Tuyere, Front.

in a horizontal fan-shaped layer (something like the flame of a gas-burner), instead of delivering it in a cylindrical or conical form.

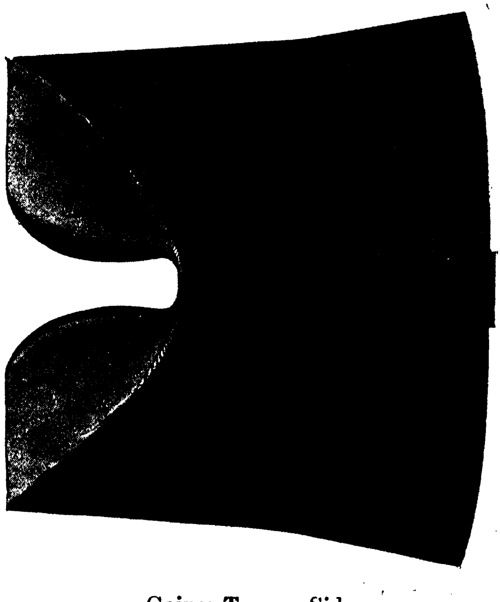
But the delivery of the blast with radial impulse in a horizontal plane is not the only result sought by this device. If that were the case, the delivery-opening might be simply a slit, as it is in the ordinary gas-burner, to which this tuyere has been compared. It will be observed, however, in the figures herewith given, that the opening varies regularly in vertical diameter, presenting at no point a perfect parallelism of

FIG. 2.



Gaines Tuyere, Back.

FIG. 3.



Gaines Tuyere, Side.

top and bottom. This peculiar symmetrical form is declared to be the result of calculation and experiment, and to effect the end for which it is designed, namely, the delivery, at each point of the nozzle-opening, of a volume of blast proportional to the field of furnace-area to be supplied by the radial blast-current from that point. The field for each tuyere is taken to be a triangular section of the furnace-area at tuyere level, bounded by radii drawn to the center of the area, midway between the tuyere and its adjacent neighbors; and the field for each special portion of nozzle-delivery (say 1 inch of the horizontal length of nozzle-opening) is taken to be the triangle formed by radii from the nozzle-center, terminating at the boundaries, thus determined, of the total field for that tuyere. In other words, the problem is treated as if the circular furnace-area at tuyere-level were divided into sectors by impermeable partitions, and each tuyere were required to furnish blast for its particular sector; each sector, moreover, being similarly subdivided into triangles having their common apex, not at the center of the furnace but at the center of the nozzle, and each horizontal inch of the nozzle-opening being required to furnish blast for its own triangle of sector-area. Obviously, the section of blast issuing from the middle of the nozzle-slit will have to supply a field extending to the center of the furnace, while the sections on both sides of it will have to serve fields of diminished area, limited by the boundaries of the sectors served by adjacent tuyeres. Hence the nozzle-opening must have a maximum vertical diameter in the middle of its horizontal length—and in like manner the vertical dimension is determined for each part of that opening, due regard being had to the frictional effect of smaller diameters, to the effect of the residual area along the furnace-walls (not directly supplied with blast, yet taking it, all the same), and to other conditions of practice. The result of a due consideration of all these elements is declared to be the form of nozzle-opening adopted in the Gaines tuyere; and this form, it is claimed, need not be varied in practice, provided the suitable size be used, that is, provided the proper relation between total tuyere-area and hearth-area be observed.

This proposition seems to me worthy of some further consideration. As a matter of fact, there are no such impermeable partitions between or within the sectors of hearth-area, as the

theory of the Gaines tuyere assumes. The normal result of the blast from the tuyeres is the production of a general *plenum* of pressure throughout the stock; and, under this pressure, the blast and the resultant gases are supposed to penetrate to all parts of the descending column of stock. Such a normal distribution is often defeated by irregularities in the stock itself, furnishing channels of superior ease to the ascending gases, and thus giving rise to premature fusions, accretions, etc. But these evils originate above the tuyere-zone, and how much they depend upon the original distribution of blast is an open question. It might be argued that if the blast be strong enough to penetrate to the center of the hearth, it will certainly distribute itself with sufficient uniformity to penetrate all available openings in the column above. And it might be added that the accretions on the hearth-wall between tuyeres, due to the failure of the blast from ordinary nozzles to reach that space, do no particular harm. The shape assumed by the crucible, from tuyere-level down, is of little consequence, so long as the interior space remains large enough to hold the molten iron, and does not become so large as to permit the iron to break through the weakened walls.

On the other hand, the establishment of a general interior pressure, such as our ordinary practice assumes, must certainly be the result of mutual reactions and adjustments concerning which we have no data. We do not know, in particular, how long it takes to establish this general equalization of pressure, or how far above the tuyeres it is established. There can be no doubt that by introducing blast through circular openings we create an irregular distribution which must be left to adjust itself somehow according to pneumatic laws. It is unscientific to assume that this adjustment will take place without affecting at all the working of the blast-furnace, and that, while recognizing the importance of the distribution of stock, we can ignore entirely the distribution of blast. And it is quite reasonable to suppose that, by introducing the blast in such a manner as to reduce to a minimum the necessary general equalization of interior pressure in the furnace, we may be able to remove one cause (whether it be a principal or only a subordinate one) of irregularities in furnace-work.

These considerations are as applicable to the proposition of

increasing the number of tuyeres as to the particular device to which I am calling attention. I would only say, in this connection, that if it be sought to secure the effect of multiplied tuyeres by modifying the form of the nozzle, without increasing the number of tuyeres, I know of no better way to determine the form of nozzle required than that which Mr. Gaines has adopted, notwithstanding the fact that it involves an arbitrary assumption as to the interior subdivisions and conditions of the furnace.

With regard to both propositions—that of more numerous tuyeres and that of modified nozzle-forms—I may be permitted to express the opinion that, while they are entirely reasonable in theory, their quantitative value must be determined in actual commercial practice. This statement is so true as to be trite, concerning all new inventions or improvements; but I think it derives some special force, in this instance, from the fact that the evils which it is sought to remedy are not wholly (perhaps not even chiefly) due to the cause which it is proposed to remove.

It follows, moreover, that the verdict of practice itself may be slow in coming, and lacking in unanimity or conclusiveness when it comes, and may require much critical scrutiny and valuation to fix its real scope and weight.

For myself, I should not be surprised if these proposed improvements should ultimately turn out to be real, but much less important commercially than their advocates have imagined.

Let it not be inferred, however, that these or any other inventions are to be, on that ground, condemned. So far as human foresight goes, the day of revolutionary improvements in the production of iron and steel is past. We must look henceforward to small economies, rejoicing as much over a dime as we used to rejoice over a dollar, and valuing every small new gain as turfmen value the fraction of a second in a racing-record. Times of low prices and fierce competition, like the present, emphasize the value of technical improvements which, in the rush of general and ample prosperity, are despised or neglected.

Moreover, I venture to point out that, in these days, the inventions (and also the improvements in the details of practice,

which have, not infrequently, more real importance than formally-announced "inventions," though they be not ranked as such) cannot be, as a rule, thoroughly tested and accurately valued at one place, and under one set of conditions. A comparison of the results obtained under varied conditions is essential to a safe conclusion.

After an experience, as a consulting engineer, of thirty-five years, during which I have been repeatedly the recipient of confidential communications concerning secret details of practice, I do not hesitate to say that, in my judgment, a free interchange of technical experiences would have been, in all cases which I can now recall, more advantageous than the attempt to preserve in silence a supposed "trade secret." I remember one instance in which the managers of three separate and rival establishments informed me, in strict confidence, what they were doing at a certain stage of the manufacture in which they were all engaged. They were all doing practically the same thing, which every one of them could have done better if they had only "compared notes;" and each was incurring considerable extra expense to keep a "secret," the substance of which his two rivals (and probably many other persons not known to me) possessed and were using already.

When the interchange of views and experiences takes place through a medium like this Institute, the advantages of a frank disclosure are multiplied. He who communicates to such a body the results reached by one, may receive in return the criticisms, suggestions and parallel results of many, who have been, perhaps unknown to him, working in the same field. It is my honest opinion, based on long observation, with exceptionally favorable opportunities, that in the vast majority of cases he gets more than he gives.

In my endeavors, as Secretary of the Institute for the past sixteen years, to secure contributions from members in practice, I have encountered two chief obstacles. The first has been the policy of business "secrecy" (in most cases, as I believe, a mistaken one), concerning which I will say no more at present, except to observe that, in most cases within my knowledge, it has not been adopted by members themselves, but is enforced upon them by their employers. Of course the Institute respects such a prohibition, however unwise or unnecessary it

may seem to be; and no contributions are accepted in violation of such a rule.

But the second obstacle to which I have referred has been, I think, more frequently encountered. It is the disinclination of many members to communicate incomplete or partial or preliminary results. They are going to write, "some day," a thorough and monumental paper; and meanwhile they prefer to communicate nothing. Far be it from me to disparage the elaborate treatises which constitute so large a part of the glory of our *Transactions*. Nevertheless, I must frankly confess that, of the numerous members who, during the last sixteen years, have declined to furnish modest contributions from experience because they meant, "some day," to contribute something more important, not one, so far as I can recollect, has ever fulfilled that expectation. Many of them have died, and what they knew, but postponed telling, has died with them. Many of them have assumed new positions and duties, and the investigations they had planned have been abandoned. Many of them have found themselves forestalled by other investigators, who began later, but were more prompt to publish. All that such deservedly outstripped wayfarers can do is to try to convince an incredulous world that, at some earlier day, they really did mean to start upon the road, or did start, but never registered their names at any point *en route*.

It is the difficulty to which I have just referred that I have encountered in a wide correspondence with members, undertaken for the purpose of securing contributions of fact and opinion concerning the special subject of this discussion. Many are not ready to express an opinion, even as to their own practical experience. Nearly all "wish to hear from others," forgetting that others similarly wish to hear from them. Almost none of them seem to recognize the value of preliminary, incomplete and indecisive reports of practice, as laying the foundation for a thorough inquiry.

I trust that this appeal may accomplish something towards the removal of such unnecessary and unwise reserve, and elicit some additional and pertinent contributions.

Mr. Edgar S. Cook, Pottstown, Pa.: I have had no experience with multiple tuyeres, but some three years ago we tried

in our furnace at Pottstown a set of triple tuyeres with three openings. We had been using seven 6-inch tuyeres, in place of which we gradually substituted triple tuyeres; I think the center-opening was 4 inches and the side-openings were 3 inches in diameter. They were directed radially. The experiment was tried with a view of getting some data as to whether it would be worth while, in a contemplated reconstruction of the furnace, to put in multiple tuyeres; and in consequence of the experiment with the three-opening tuyere we decided not to go to that expense.

The first result of putting in the three-opening tuyere was rather beneficial. The furnace seemed to make more iron and to drive a little better, and we felt quite encouraged; but afterwards we fell back into our normal work with no apparent benefit at all. It made no difference at all, over a longer period, whether we used the single tuyere or the three-opening tuyere; and for that reason we did not go to the expense of changing our plans to put in the multiple tuyeres which were coming to be more or less talked about at that time.

I think a good many furnace-managers may have realized an immediate benefit from increasing the number of their tuyeres, particularly when the additional tuyeres have been put in during the blast (that is, when the furnace has been stopped for a few days and the extra tuyeres have been put in).

But it seems probable to me that this good result has been due to the same causes which made our use of the three-opening tuyeres apparently advantageous at first, namely, accretions on the walls were melted away and temporarily the product of the furnace was increased. But after a while the product fell off to the usual average, with the disadvantage of working, as Mr. Hartman said, more on our bosh-wall. The furnace was old, and as we had a number of breakouts, the brick-work between the tuyeres giving way, we discontinued their use.

Later we tried the triple tuyeres again, in the early part of the present blast, and found no benefit from them at all. I imagine that a good deal of the improvement that has come from the increase of tuyeres has been in cases where the tuyeres have been deficient in number, because the area of the crucible has been increased without proportionately increasing the number of tuyeres. In cases where the tuyeres have been fairly

well adjusted to the diameter of the crucible, increasing the number has not proved to be of any particular benefit.

WM. B. PHILLIPS, Pittsburgh, Pa.: I have had some little experience in this matter, particularly with the southern furnaces, which, as is well known, are running on stock quite different from that used in the North, and liable, moreover, to considerable irregularity in composition and physical condition.

Some of the large furnaces in Alabama changed from eight to sixteen tuyeres, and one of them put in twenty-four tuyeres. The change increased the output, but in many instances did not improve the quality of the iron. In fact, the proportion of foundry-grades was not so high with the larger number of tuyeres. The 24-tuyere furnace has since gone back to sixteen, not having realized good results from the larger number.

In one case a furnace changing from eight to sixteen tuyeres stripped the lining down to the jacket within three weeks, part of the blame being laid on the brick and part on the practice.

It is of interest in this connection to note that, in the production of "basic" iron, by far the greater amount and the better quality has been made in a furnace with the usual number of tuyeres. So far as I know, there has been no systematic effort to ascertain the general effect of the multiple tuyere upon the quality of the iron as determined in the laboratory. Under such circumstances yard-grading is unreliable; for it is well known that the iron may have changed in composition without showing a corresponding change in appearance. The economy of the multiple tuyere is to be measured, not only by increased production, but also by the effect upon the quality of the iron.

RALPH H. SWEETSER, Everett, Pa.: Some of our up-to-date furnace-men seem very modest about giving their ideas on the question of multiple tuyeres. There is one thing that can be told, namely, that the Everett furnace, in changing from eight to twelve tuyeres this last summer, taking the total area of the eight tuyeres and putting the same area into twelve tuyeres, necessarily giving smaller tuyeres, could not get as good results. We had to bring the total area of the twelve tuyeres down below the total area of the eight tuyeres which we had in the first place, to get the same results as we got before. The furnace

is 75 feet high; the hearth is 11 feet in diameter, having been increased from 10 feet 6 inches with the eight tuyeres. The trouble with the twelve 4½-inch tuyeres is to get the proper penetration of the blast; and as yet I see no advantage over the eight 5½-inch tuyeres.

J. M. HARTMAN, Philadelphia, Pa.: To make clearer, perhaps, what I have said in my paper as to the use of oval tuyeres, let me suppose that a furnace has eight columns and eight round-nose 6-inch tuyeres. If greater side-dispersion of blast is desired, it is not necessary to introduce sixteen tuyeres. Eight oval tuyeres, properly proportioned to the required volume of air, will effect this result without weakening the walls. They can be readily adapted to the existing tuyere-breasts and embrasures; and all desired side-dispersion can be secured by giving proper proportions to the nose.

O. S. GARRETSON, Buffalo, N. Y.: Although I have had no experience with blast-furnace tuyeres in smelting ore, I have had some experience in melting iron in a cupola, and I have been led to make some noteworthy changes in tuyeres. I have a furnace now working in Buffalo which has two tuyeres, 12 by 18 inches in section. The furnace is about 5½ or 6 feet in diameter, and has worked ten or twelve years to perfect satisfaction. Before putting in tuyeres of that kind it had a tendency to scaffold—would very often be scaffolded over—and we would have trouble to drop the bottom and let the charge come down. In studying the matter carefully I observed that the blast concentrated through a small tuyere blew out the fire on the stock and cooled the iron that lay in front of the tuyere, and that scaffolding began there and finally continued clear to the outside; and I thought that if I increased the size of the tuyere it would not have a tendency to blow out the fire. As a result, we put in 12- by 18-inch tuyeres, and the furnace worked perfectly. I do not think we have ever had a scaffold in that cupola since those tuyeres were adopted. We employ the ordinary cupola blast-pressure from a No. 7 Sturtevant fan. My theory is that the force of the blast, when it strikes the fuel, is distributed; it goes down and up; and penetration to the center of the furnace is not due to the initial direction of

the air, but simply to its volume and distributed pressure. We give a sufficient volume of air to that furnace; and the current does not go straight to any one particular point or along one line, but, looking into the tuyere, one sees the fuel at all points glowing and burning. There is no such special concentration of air on one spot as actually to extinguish the fire in that spot.

A. E. Barton, Supt. Tenn. C., I. and R. R. Co., Ensley, Ala. (communication to the President): The Tennessee Coal, Iron and Railroad Company was the pioneer of the introduction in the South of an increase in the number of blast-furnace tuyeres. In December, 1896, No. 1 furnace at Ensley was changed from eight tuyeres to sixteen. This furnace had been in blast, previous to the change, a little over two years and a half, and immediately before the change was made had been working more or less unsatisfactorily, owing to dust-accumulations on the boshes. The furnace worked under excessive pressure when the stove-heats were forced, and scaffolded and slipped periodically.

It was decided to increase the number of tuyeres in order to make the furnace work more on the boshes. The furnace was banked; the old cast-iron tuyere-jacket, with eight tuyere-openings, was removed, and a new jacket with sixteen openings was placed around the old brick-work, and filled in behind with ganister. New holes were then cut in the brick-work and sixteen bronze coolers and tuyeres were inserted, extra connections being made with the bustle-pipe. The furnace was started up without trouble six days after being banked. The volume of blast had been formerly about 30,000 cubic feet of air per minute; and this amount was kept approximately the same after starting up. The blast-pressure, however, was reduced from an average of 12 pounds to about 8, and the furnace drove freely under this pressure with high heats, and showed no signs of scaffolding or slipping, the grade of iron being much more uniform. The first two months' work with sixteen tuyeres showed marked improvement over the previous work with eight; the output being increased 27 per cent. and the coke-consumption decreased 12 per cent. This gain, however, was doubtless much more apparent than real; for about 1100° Fahr. had been

the average heat carried on the stoves when working eight tuyeres (a higher temperature of blast having been found to cause irregularity in the working of the furnace), whereas 1300° and more was carried when working sixteen tuyeres, without interfering with regularity of working. This furnace is in blast now (July, 1898), and has been in blast continuously since the change was made, having made upwards of 270,000 tons of iron on her last lining, with an ore-mixture yielding 36 per cent. of iron in the furnace.

These results were considered so satisfactory that in March, 1897, No. 3 furnace at Ensley was banked and changed to sixteen tuyeres in the same manner, the results of the first month's working, after the change, being almost equally satisfactory. When the change was made on this furnace extra stove connections were put in, which allowed the use of an extra stove on each furnace by utilizing the stoves on the fourth furnace at Ensley, which was not in blast.

Early in June, 1897, No. 2 furnace at Ensley was banked, and twenty-four tuyeres were put in, the coolers having only 2 inches of space between them at the nose. An entire circle of brick-work was cut out around the furnace; the cast-iron tuyere-jacket with twenty-four openings was set up in place; the bronze coolers were inserted, making a metallic joint with the jacket; the tuyeres and blast-connections were put in place, and the furnace was started up. The slag formed, running between the bronze coolers and chilling there, lined the wall of the upper part of the crucible. The results from this furnace were not quite so satisfactory as with the two having sixteen tuyeres, though they showed improvement on the previous working with eight tuyeres. By reason, probably, of insufficient penetration of the blast, the furnace would not make any large proportion of high-grade iron, except under what was considered an excessive volume of air—in fact, would not make foundry-iron at all with less than 36,000 cubic feet of air per minute. The nozzle-openings of the tuyeres, which were 6 inches in diameter, were reduced to 4 inches, and afterwards to 3 inches, with some benefit, though the results obtained were not so satisfactory as with the furnaces having sixteen 6-inch tuyeres. In November, 1897, twelve tuyeres were removed, and this furnace is now working on basic iron with

twelve tuyeres, having 7-inch openings. The change from twenty-four to twelve tuyeres curtailed the output, but slightly lowered the fuel-consumption when making foundry-iron. This was doubtless due to the reduction of the volume of air from 36,000 to 28,000 cubic feet per minute. The furnace, however, still occasionally has dust-accumulations formed on the boshes which give more or less trouble, though not to the same extent as when working on eight tuyeres.

The furnaces with sixteen tuyeres appear to have absolutely clean boshes, and have given no trouble with regard to hanging and slipping since the sixteen tuyeres were put in. Their boshes, however, require to be well protected by water-cooling devices.

No. 4 Ensley has recently been put in blast with sixteen 6-inch tuyeres, having thirteen rows of bosh-plates 13 inches apart, and has started away very satisfactorily. The Alice furnace of the Ensley division was also equipped with sixteen 6-inch tuyeres, and has been working very satisfactorily on "basic" iron.

Mr. Means, Superintendent of the Sloss Iron and Steel Company, has given the increase of number of tuyeres a trial at the North Birmingham furnaces, and also Mr. Dowling, Superintendent of the Tenn. Co.'s furnaces at Bessemer, tried sixteen tuyeres on one furnace. These are all somewhat smaller furnaces than those at Ensley; and I understand that the experiments were not altogether satisfactory. The Pioneer Mining and Manufacturing Company, at Thomas, Ala., has sixteen tuyeres in one of its furnaces.

To sum up, it would appear from experience at Ensley that large furnaces, using a soft coke, and liable to have dirty boshes, are benefited by an increased number of tuyeres, as the furnaces work more regularly, carry higher heat without disturbance of working, and for this reason give a larger output and lower coke-consumption, measured over a considerable period. With a large furnace on easily reducible ore, and an effective volume of air, the quality of the iron produced seems to be the same on a large number of tuyeres as on the smaller number.

N. B. Wittman, Birdsboro, Pa. (communication to the Secretary): Our experience at Birdsboro with multiplied tuyeres

has been as follows: We have increased the number from six to eleven, and decreased the diameter of nozzle-opening from six inches to four inches. It is exceedingly difficult to make perfectly fair comparisons of the results, for the reason that the proportions of coke and anthracite in our fuel are varied from time to time, and, although our ore-mixture has been essentially uniform, there have been minor changes which may have been not without influence. Moreover, we have increased our boiler-power, and have aimed to make the plant more efficient generally. The most important change we have made was undoubtedly that of the tuyeres, and I think it has been beneficial. Comparing the work now done with that of the former blast, and taking into consideration the proportion of coke in the fuel, the time of year, and the grade of the product, there has been a relative increase in yield and a saving of fuel.

I have not noted any counterbalancing disadvantages. We have burned fewer tuyeres than formerly. As the fuel-duty has shown a net increase, the loss of heat due to the increased volume of tuyere-water cannot be considered as commercially significant.

Although we have not realized the extreme benefits reported by some other experimenters in this line, I regard the change as a distinct improvement.

LEONARD PECKITT, President, Crane Iron Works, Catasauqua, Pa. (communication to the Secretary): Our No. 1 furnace has been working with twelve tuyeres since January 1, 1898, and, although the output has been considerably increased, I think the larger number of tuyeres may have less to do with this result than the fact that we are using an increased proportion of coke with the anthracite of our fuel.

After this furnace had been in blast about three months, the yield suddenly fell off, and the furnace began to work very irregularly. At that time we were using 5-inch nozzles, and the blast was evidently not penetrating to the middle of the furnace. Upon the substitution of 4-inch nozzles, an improvement was at once noticed, and the furnace is now (July) down to normal conditions again. I am inclined to think that for our work eight tuyeres would do as well as twelve.

By-Product Coke-Ovens in the United States.

Discussion of the Paper of Mr. Blauvelt on the Semet-Solvay Plant at Ensley, Ala. (See p. 578.)

(New York Meeting, February, 1899.)*

E. W. PARKER, Washington, D. C.: In connection with Mr. Blauvelt's paper, a brief sketch of the development of by-product coke-making in the United States may prove of interest. The first radical departure from the prevalent beehive oven to the by-product retort oven was inaugurated in 1891 by the Solvay Process Company, at Syracuse, N. Y. In that year was begun the construction of 12 Semet-Solvay ovens, which were completed in 1893. During 1893 these ovens produced 12,850 tons of coke, an average of a little over 1000 tons each. In the following year the production amounted to 16,500 tons, not quite 1400 tons per oven. These 12 ovens were in reality an experiment, the success of which was attested by the addition to the same plant of 13 more ovens, begun in 1895 and completed in 1896. They were followed by the construction in 1896 of 25 ovens of the same type at Sharon, Pa., and 50 more at Dunbar, Pa. These 100 ovens represented the total number of Semet-Solvay ovens completed and in operation up to the close of 1897. The construction of 180 more, however—120 at Ensley, Ala., and 60 at Benwood, near Wheeling, W. Va.—was begun in 1897. The plant for the latter is laid out for double the number stated. The 60 ovens begun in 1897 have now been completed.

An initial plant of Otto-Hoffman ovens was built at Johnstown, Pa., in 1895, and began making coke in 1896. One hundred and twenty additional ovens of this type were begun in 1896 and completed in 1897 at Glassport, near McKeesport, Pa. During the present year 100 Otto-Hoffman ovens are being added to the plant at Johnstown, and 400 of the same type are in course of construction at Everett, near Boston, Mass. The plant at the latter place is laid out for a total of 1200 ovens.

* NOTE.—This contribution, though delayed in its presentation until the New York meeting, is printed here, in order not to separate it from the paper to which it refers.—R. W. R.

From this it will be seen that on January 1, 1898, there were 280 by-product ovens completed and in operation, while during the year there are 680 building, the number under construction in 1898 being nearly 2½ times the total number completed at the end of 1897. The 30 Newton-Chambers ovens at Latrobe, Pa., and 3 Slocum ovens at Bolivar, which have not been operated on a commercial scale, are not included in this statement.

Reduced to tabular form, the record of the by-product coke-ovens in the United States has been as follows:

Year.	Ovens Built.	Ovens Building.	Coke Produced in Short Tons. (2000 Lbs. Av.)
1893............	12	0	12,850
1894............	12	60	16,500
1895............	(a) 72	63	18,521
1896............	160	120	83,038
1897............	280	......	261,912
1898............	280	680	....

(a). Sixty of these did not begin making coke until 1896.

Statistics of the amount of tar, sulphate of ammonia and surplus gas obtained from the above operations are not available; but Mr. S. M. Atwater, Secretary of the Semet-Solvay Company, has furnished the following statement of the results obtained from one Semet-Solvay oven, operated under normal conditions for one year:

Record of One Semet-Solvay Oven for One Year.

Coal charged into oven, ...	1,600 short tons.
Coke produced, ...	1,150 "
Total gas produced, ...	16,000,000 cubic feet.
Gas used by ovens, ...	10,000,000 "
Surplus gas available for other purposes, .	6,000,000 "
Tar produced, ...	16,000 gallons.
Ammonia produced as sulphate, . . .	16 short tons.

Assuming from the above that, for each 1000 tons of coke made, the surplus gas will average about 5,000,000 cubic feet, the tar about 14,000 gallons, and the sulphate 14 tons, the production of these substances in 1897 could be estimated about as follows:

Gas, ...	1,300,000,000 cubic feet.
Tar, ...	3,640,000 gallons.
Sulphate of ammonia, ...	3,640 short tons.

Corundum in Ontario.

Discussion of the Paper of Archibald Blue, Toronto, Canada. (See p. 565.)

ALFRED E. HUNT, Pittsburgh, Pa. (communication to the Secretary): Mr. Blue's statement (page 576) that, "owing to the presence of iron and other impurities, makers of aluminum assert that native corundum is unsuited for the production of that metal," is somewhat incorrect, at least so far as it may refer to any objection made by the officials of the Pittsburgh Reduction Co. to the Canadian corundums. We recognize corundum as among the richest and purest of the ores of aluminum, and the material furnished from Canada would offer for us the advantage of a supply near to our works for the manufacture of aluminum at Niagara Falls.

The real difficulty which we find in the use of corundum for this manufacture is the cost of the raw material as compared with that of native bauxites. In this item we include not only the price of the corundum as it has been offered to us, but also the expense of grinding it to an impalpable powder, which must be done before it can be used directly in the manufacture of aluminum, and the cost of preliminary chemical treatment for purification—which latter operation, however, is also required for bauxite.

I agree with Mr. Blue that the introduction of new and improved processes has greatly reduced, during the past ten years, the cost of manufacturing aluminum; but I am inclined to believe that any further improvements, facilitating the use of corundum for this purpose, would apply, in equal or greater degree, to the common clays (silicates of aluminum) which exist in all parts of the world, are more easily mined and handled, and are likely to be always cheaper per unit of aluminum than any corundum.

The Relations Between the Chemical Constitution and the Physical Character of Steel.

Discussion of the Paper of William R. Webster, Philadelphia, Pa. (See p. 618.)

H. H. CAMPBELL, Steelton, Pa. (communication to the Secretary): I wish to thank Mr. Webster for the copious quotations he has made from my writings, as he has given nearly all the arguments I wish to advance.

It is necessary to note, however, his statement that most of his experiments were made on universal-mill plates, and that these were not subject to my condemnation of irregularity. I think this is a mistake. It is well known that universal-mill plates are finished under very different conditions of temperature, with the express purpose of regulating the tensile strength. This is a perfectly legitimate practice; but it renders almost worthless any calculations on the relations of the chemical and physical equations when the influence of such finishing is omitted.

Another important point Mr. Webster has not answered at all. I have shown by an excerpt from his own table that two entirely different formulæ may be constructed; and it is undeniable that one is as valid as the other, since Mr. Webster started with no premise at all save an assumption that the effect of carbon was constant. This was an assumption pure and simple, and hence has no special virtue.

The whole result of Mr. Webster's investigation is not that the effect of carbon is a constant, and that those of phosphorus and manganese are variables. On the contrary, there are two answers, with nothing to show which is right. We learn that if carbon is a constant, then phosphorus and manganese are variables; but that if phosphorus and manganese are constants, then carbon is a variable. This is a distinct advance over complete ignorance, but it is not final and satisfactory from either a mathematical or metallurgical standpoint.

Regarding my own formulæ, I would say that they are used regularly in the testing-laboratories of the Pennsylvania Steel Company. The tensile strength of every heat is calculated

from the chemical composition, and the results are compared with those obtained on the testing-machine. In order to show the concordance of results, I have taken from the records the figures from the most recent 250 heats of acid steel, and the most recent 250 heats of basic steel. No heats, however, were taken that gave over 70,000 pounds ultimate strength. Of the 500 heats thus taken, it was found that on 311, or 62.2 per cent., the strength as given by the formula and the strength as shown on the machine agreed within 1000 pounds, and 89 per cent. were within 2000 pounds. Moreover, 96.6 per cent. were within 3000 pounds, 99.2 per cent. within 4000 pounds, and only one heat, or 0.2 per cent., was over 5000 pounds. On all these heats a sample test had been kept, and the 55 heats that showed over 2000 pounds variation were again analyzed for carbon, phosphorus and manganese. This retesting gave no evidence of any mixing or substitution, for the phosphorus and manganese corresponded as nearly as could be expected, but there were cases where the carbon varied three points or even more. This was not a surprise.

No scientific investigation avails much which rests on the color determination of carbon. This point I have insisted on before. It answers Mr. Webster's request for the results on the separate heats composing the groups from which my formulæ are derived. To work 1880 carbons by combustion would take a chemist nearly a year. It would be the best way to do if some one would pay the chemist, and if some one would also do the extra mathematical work required by the Method of Least Squares. Color determinations on low steels are useful as a check; they are somewhere near right, but as a basis for accurate scientific work—as a foundation for learned and laborious discussions on the influence of the elements—they are very unsatisfactory and inconclusive.

In case of the 55 heats above mentioned, the carbons were all repeated by color. In cases where there was discrepancy the test was made again, and the color was compared with two different standards, and in cases of doubt a combustion determination was made. No attempt was made to pick out the result that suited the formula best, but in each case an average was taken of all the results. This was not strictly scientific, but it seemed fair and practical. The ultimate strengths were

then calculated from the formula according to the new chemical determinations, and the results are given below:

Difference Between the Tensile Strength as Found by the Formulæ and as Shown by the Testing-Machine.

Acid Heats. Total Number of Heats = 250.

	No. of Heats.	Per Cent.
Difference of less than 1000 pounds,	149	59.6
Between 1000 and 2000 pounds,	88	35.2
" 2000 " 2500 "	7	2.8
" 2500 " 3000 "	1	0.4
" 3000 " 3500 "	1	0.4
" 3500 " 4000 "	3	1.2
" 4000 " 5000 "	0	0.0
More than 5000 "	1	0.4
	250	100.0

Basic Heats. Total Number of Heats = 250.

	No. of Heats.	Per Cent.
Difference of less than 1000 pounds,	162	64.8
Between 1000 and 2000 pounds,	77	30.8
" 2000 " 2500 "	4	1.6
" 2500 " 3000 "	0	0.0
" 3000 " 3500 "	3	1.2
" 3500 " 4000 "	2	0.8
" 4000 " 5000 "	2	0.8
More than 5000 "	0	0.0
	250	100.0

By looking at the last column of this table it will be found that 59.6 per cent. of the acid heats came within 1000 pounds, 94.8 per cent. within 2000 pounds, 98.0 per cent. within 3000 pounds, and 99.6 per cent. within 4000 pounds. In the basic steel, 64.8 per cent. came within 1000 pounds, 95.6 per cent. within 2000 pounds, 97.2 per cent. within 3000 pounds, and 99.2 per cent. within 4000 pounds, and all within 5000 pounds.

It would be very interesting to know what caused the one poor lonely acid bar to show such different results from his fellows, but every one acquainted with the casting and rolling of test ingots, and the measuring, testing and recording of the tests in a busy steel-works, will be able to think of some possibilities that may account for the discrepancy.

William R. Webster, Philadelphia, Pa.: I do not at all agree with Mr. Campbell that great differences in finishing-

temperature are used in rolling universal-mill plates, as in all well conducted mills they endeavor to finish at a uniform temperature.

If Mr. Campbell can construct from my tables a formula in which the carbon is made a variable and one of the other elements a constant, and get as good results, he will add to our knowledge on this subject.

Carbon has for many years been the one element that all relied on in grading steel, and it has also been known that low-carbon steels could contain more phosphorus than the high-carbon steels without being rendered brittle, this being very marked in the higher tool-steels. Up to this time I cannot see any reason for giving carbon a variable value for steel of 70,000 pounds and under; but as this investigation is continued it may be shown that better results can be obtained by doing so.

The values used for each element are those which gave the best results after many trials on over 1000 test-pieces used in this investigation, besides hundreds of trials on the steel graded in the course of every-day routine work at the mill.

The color-carbons were used, as is the practice at all steel-works in their usual work. Of course it is better to use the carbon-determinations by combustion, as Mr. Campbell states, *but I cannot agree with him in first grouping a lot of the tests together and taking part of the drillings from each test-piece for the carbon and an average of the other elements for the group*, as was done by Mr. Campbell in all of his investigations. This, as before stated, gives a more constant relation between the elements than is found in the individual test-pieces.

In Mr. Campbell's work, as before shown, the value of any one of the elements—carbon, phosphorus, or manganese—and the value of pure iron seem to depend on the number of tests considered. If the effect of each element is constant and not dependent on the amount of the other elements present, this should not be the case. There seems to be a disturbing element running all through the successive values arrived at, and strongly indicating that the values of carbon, phosphorus and manganese are not constant under all conditions of the series of tests, in either the acid or the basic steels considered.

I do not think the results obtained justify Mr. Campbell in taking the strong stand that manganese does not affect the

ultimate strength of acid open-hearth steel until it is raised to over .60 per cent. In this class of steel he uses 1210 pounds for each .01 per cent. of carbon, instead of 950 pounds, as in the basic steel. This increase of 260 pounds for each .01 per cent. of carbon represents an increase of 27 per cent. in its value, and it includes all of the effect of the manganese and part of the effect of the phosphorus. This is indicated by the lower value of 890 pounds instead of 1050 pounds for phosphorus, being a reduction of 160 pounds for each .01 per cent., or a reduction of 15 per cent., and the manganese is not considered at all, being a reduction of 85 pounds for each .01 per cent. of that element.

I do not mean to question the accuracy of Mr. Campbell's work in using the method of least squares in arriving at these values, but I do claim that his results indicate strongly that the amounts of the elements present have an indirect effect on each other, and that this is shown by the different values arrived at for each element, depending on the number of tests considered.

If Mr. Campbell's position is correct in that, the method of least squares would not give intelligible results, when he considered the effect of copper, silicon and sulphur with the other elements, on account of the copper, silicon and sulphur having very little effect on the ultimate strength of the steel. It certainly casts a reflection on the applicability of this method to the problem in hand, as it presupposes a knowledge of the effects of three of the unknown quantities under consideration.

I claim that as we know that manganese has an effect on the ultimate strength of steel, we should always allow for it directly, instead of allowing for it indirectly by giving carbon a larger value, which only holds good when there is a fixed relation between the carbon and manganese present. The steel-works now use different methods of recarburization; and as this investigation is continued, it will be found necessary to allow for the effect of manganese in both acid and basic steel, whether made in the open-hearth furnace or in the Bessemer converter.

I congratulate Mr. Campbell in that he finds such a close relation between the estimated ultimate strengths and the actual strength of the two by three-eighths bars rolled from the small test-ingots rolled under uniform conditions of temperature. No doubt the tests of the finished material give

correspondingly good results, although he has not said so. Mr. Campbell says, on page 7 of his *Structural Steel:*

"We discovered many years ago that we had been running with an error of .11 per cent. in all our low-carbon determinations and .13 in all the high steel. Thus, steel of .09 carbon had been regularly determined as .20, and .50 carbon as .63. Customers ordered steel, found it right, or found it too hard or too soft, and ordered the next lot accordingly. Years had rolled by, and every customer knew just what he wanted, and could learnedly discuss the special nature of .64 and of .76 carbon. The discovery of the error in the standards was a rude shock, and the change to the new order of things was the work of many months, and a diplomatic catering to prejudice, mixed with a very strong disinclination to an open acknowledgment that we had been altogether wrong."

This is a fair illustration of how little attention was formerly paid to the relation between chemical composition and the ultimate strength of steel by Mr. Campbell, the manufacturers, or anyone else—whereas, to-day, some of our mills are rolling the steel into the finished product altogether from the estimated ultimate strength based on chemical composition of each heat. They do not make any preliminary tests at all, and the only tests are made on the finished material, after the whole of the heat has been rolled. The steel is never allowed to cool, from the time it is cast until it leaves the rolls as finished angles, bars, etc. This great advance in the method of working means a yearly saving in the coal-bill of thousands of dollars, and another large saving in the cost of handling the material.

In order to study more closely the relations between the estimated ultimate strengths arrived at by my method and those of Mr. Campbell and Mr. Cunningham, I have constructed tables on the same general plan as Table No. XI., giving the values of all three observers in each table; the carbons being in all cases from .06 to .28 per cent. inclusive. (My values are for open-hearth steel, and sulphur has not been considered, but it can be used in connection with these tables by adding 500 pounds for each .01 per cent. of sulphur above .065 per cent. sulphur and deducting 500 pounds for each point below this amount from my values given.) In the first of these tables the manganese is zero in all cases, and for that reason the values of Mr. Campbell's acid steel and Mr. Cunningham's values are higher than the others, as in these cases the value of manganese has been indirectly included in that of the carbon by Mr. Campbell and Mr. Cunningham. But taking the other tables, with manganese .30, .40, .50 and .60 respectively, the estimated

ultimate strength of one observer agrees with the others much more closely than one would suppose from the different values used for each element.

There are also six tables having carbons .29 to .51 inclusive, with manganese .50, .60, .70, .80, .90 and 1 per cent. In all other respects they are the same as the other tables mentioned above. Of course, with the higher carbons these values may not apply, but the sooner we try them the sooner we shall know what modifications will be required.*

I do not think that any of us, up to the present time, have been fortunate enough to arrive at the right values for the different elements, and I doubt very much if we shall ever have a simple mathematical formula to express these values, as the elements present have, no doubt, an effect on each other. It will, of course, take much time and work to decide on the best modifications of the values and methods now in use.

R. W. RAYMOND, New York City: In the discussion of Dr. Dudley's famous paper on "The Wearing Capacity of Steel Rails, in Relation to their Chemical Composition and Physical Properties" in 1881, I presented an attempt to deduce from his tables,† by the method of least squares, some indications as to the effect of certain elements in rail-steel. At that time I was somewhat more confident than I now am, of the value of that method in the discussion of such observations; but I pointed out one fundamental objection to it, which I will now state again, namely, that it involves an arbitrary assumption as to the form of the functions into which both constants and variables enter. After eliminating all the effects of other conditions, it is still necessary to assume something as to the law of the effect of chemical constituents; and the universal assumption has been, that the general formula is a purely arithmetical one: that the effect of each element is in direct proportion to its quantity; and that the total of all such effects is expressible in the formula $Ax + By + Cz$, etc., in which A, B and C represent the percentage of the element present, and x, y and z are constant multipliers, each empirically determined for the corresponding element. Some of the multipliers may be negative;

* Copies of these tables have been filed at the office of the Secretary, for examination by any who may be interested.

† *Trans.*, ix., 605.

but the algebraic sum of the products thus obtained is assumed to represent the total effect. It is highly probable, *à priori*, that the law of nature is much more complicated than this. If the effect is to be measured, for instance, in terms of tensile strength, it is highly improbable that a general law exists, according to which so many units of carbon, multiplied by a certain modulus, will add so many pounds per square inch, and twice as many units twice as many pounds. Such an assumption may be practically valuable for a particular range of practice and quality of product, but there is no proof that it obtains as a natural law —on the contrary, there are many results of experiment which indicate the contrary. It seems to me that all the investigators in this line are seeking and getting merely empirical formulas, which may be of the highest value in the laboratory and the mill, without having any value at all as scientific inductions. In practice, their relative usefulness must be tested by the closeness with which, within the range to which they apply, they approximate the results of physical tests. From this standpoint, it seems to me, theoretical criticisms amount to little. If, in a given set of tables constructed upon such formulas, carbon is "sometimes a constant, and sometimes a variable," I do not see how that fact can count for much, provided the tables give practical satisfaction in use. Probably all the elements involved in these tables are variables, in that sense.

But for that comparison of results which will enable us, both to arrive at the best empirical formulas, and, perhaps, ultimately at a general law, the fundamental requirement is that experimental data shall be accurately determined by uniform methods, and fully and intelligently reported. A vast amount of statistical misinformation still encumbers our technical literature. They say that when our government established the Signal Service weather bureau there were in the Smithsonian Institution tons upon tons of absolutely worthless meteorological reports, undigested and undigestible, furnished from all over the world by old gentlemen who looked at their thermometers every two hours, day and night, and reported the result with utterly wasted diligence. We are not quite as badly off as that, but we do need data capable of discussion; and such work as Mr. Webster, Mr. Campbell and others have done, together with the work conducted by international committees in this field, will give us what we need.

Modern Cupola Practice, with Special Reference to the Discussion of the Physics of Cast-Iron.

Discussion of the Paper of Bertrand S. Summers, Chicago, Ill. (See p. 396.)*

R. S. MacPherran, Milwaukee, Wis. (communication to the Secretary): The importance of carbon in pig-iron, as well as the desirability of buying, for some purposes, irons which are high in carbon, has always been recognized. Mr. Summers, I think, has given undue weight to the carbon-determinations. The combined carbon often decreases materially as the center of the pig is approached, with a corresponding increase in the graphite. Even the determination of total carbon, to be of service, must be supplemented by that of other elements. By it alone, the softness of a foundry-iron cannot be determined. Car-wheels and No. 5 iron, for instance, frequently contain 3.75 per cent. of carbon. This is the percentage specified by Mr. Summers for his No. 1 iron, "with the silicon about anywhere the furnace-man wants it." It is true that, other things being equal, the higher the carbon the softer the iron for foundry-use. But to introduce any given percentages of graphitic and total carbon into a pig-iron specification would but make the specification more complicated, without increasing its value in a commensurate degree.

Mr. Summers does not believe in the governing of mixtures by the silicon, and, in support of his views, gives several tables. It seems to me that both his conclusions and his tables are somewhat unsatisfactory. He comments on the irregularity in carbon, but fails to note the significance of the irregularity in silicon. The variation of 0.7 per cent. in silicon, for example, is too great to be accounted for by difference in blast. It might better be attributed to bad grading at the furnace, unsystematic sampling, or irregular charging. Deductions drawn under such conditions are of little value.

In ordinary work the castings from the same mixture should

* Secretary's Note.—A part of this discussion is the paper by F. E. Bachman on "The Silicon-Control of Carbon in Cast-Iron," published page 769.

not vary 0.15 per cent. in silicon, and for cylinders and special work do not vary 0.10 per cent. As the percentage of silicon increases from the least possible amount up to nearly 3 per cent., the ratio of graphite to combined carbon increases, *if the other conditions are constant*—that is, if the conditions of cooling are not altered, and if silicon and iron are the only elements, the percentages of which vary. The influence of any given percentage of silicon is the more powerful, the lower the total percentage of that element present. A change of 0.10 to 0.20 per cent. in No. 5 iron makes a decided difference in the carbons; whereas, in a No. 1 iron, this effect is often overshadowed by the more potent influence of the cooling-conditions. To the latter Mr. Summers seems to have paid no attention. I say "seems," because in no place does he mention them, or record any attempt to preserve uniformity of casting and cooling in his test-pieces. It is just possible that some of the anomalies in his Tables I. and IV. are due to this cause. The conditions of pouring, cross-section of piece, etc., are of the utmost importance in determining the relative amounts of graphitic and combined carbon in the casting. An iron which, in a small casting, would be white, may be perfectly gray in a large one. It may, indeed, be said that all iron is gray, if slowly enough cooled. Let me illustrate this proposition in physical as well as chemical terms. An iron cast in a 1-inch test-bar may give a tensile strength of 35,000 pounds to the square inch, while, in a heavy fly-wheel segment, it will not average 20,000 pounds.

In conclusion, I wish to express my agreement with Mr. Summers that the indiscriminate use of ferro-silicon is to be condemned. Within its proper sphere, it is often of great value; but it is too frequently employed as a remedy for all ills.

Dr. R. Moldenke, Pittsburgh, Pa. (communication to the Secretary): In studying Mr. Summers's paper one must admit the patience with which he has worked to present a theory at variance with the conclusions of all the leading scientific founders; but one cannot fail to notice the continual contradictions, or, let us say, a periodic hiatus between his results and the conclusions derived therefrom. He frankly discards the idea that silicon is the important element in cupola-mixtures, and aims at controlling the graphite. Right here Mr. Summers

will be met with the stubborn fact that the amount of graphite present in a casting is a function of the silicon present, the casting-temperature, and the rate of cooling; the total carbon, of course, coming into play to some extent, though it is very seldom so low that special precautions (in the form of an increase in silicon by the way) must be taken to keep the casting safely fit to be machined. Leave out the silicon, and how would Mr. Summers govern the graphite? The hotter the casting-temperature, the higher the combined carbon, which means lower graphite; the thicker the section, the slower the rate of cooling, and hence the higher the graphite. As the rate of cooling depends upon the dimensions of the casting, which cannot be changed, and as the casting-temperature is capable of being lowered only by allowing the ladle to stand after tapping, and cannot be raised if required, what can Mr. Summers do if he ignores the silicon?

A much more satisfactory way is to adjust the silicon-contents to the fuel-ratio, blast, and other cupola-conditions, and thus approximate correct results every time. The instances cited by Mr. Summers of pig-irons in which the silicon was found not to govern the graphite are entirely misleading, as it is well known that furnace-casts vary very much in temperature, and the same iron, if cast hot, may be white, while, if cast cold, it will be gray. This proposition applies to cases of comparatively low silicon. As the percentage of silicon rises, the sensitiveness of the carbon is lessened to some extent; but the rule holds true, nevertheless.

Considering *seriatim* the points brought out by Mr. Summers, we find, first, that he admits the most prominent feature of the effect of silicon to be the promotion of the graphite-formation. He claims that as to the requirements of this reaction nothing absolute can be stated, which is not quite the case; for many experienced founders govern their work by the silicon-contents exactly. Again, the claim is made that charcoal- and coke-irons of the same composition show differences due to carbon-conditions, in which temper-carbon plays an important part. While this is an open question, it is doubtful whether temper-carbon will be found in any cast-iron not subjected to prolonged annealing. The effect of greater oxidation in coke-irons has much to do with the peculiar appearance of charcoal- and coke-iron fractures.

The question of sulphur is taken up next, and its effects are noted. The limit proposed by Mr. Summers, if applied to ordinary foundry-work, is too high, though special kinds of castings may stand 0.10 per cent. of sulphur without harm.

On the manganese-question it will pay us to "go slow," in view of the tendency of this element to harden the metal, and (by combination with sulphur to make hard spots) to cause trouble in the machine-shop.

It were better if Mr. Summers would avoid the term "semi-steel," which means nothing, as the metal it is used to designate is only a cast-iron running very low in carbon, and having a high tensile strength (below that of malleable cast-iron, by the way), a little more elongation than the weaker grades of cast-iron, and, as he states, more magnetic permeability. This is no reason why a new name should be used, in these enlightened days, for the upper grades of material having all the properties of cast-iron and none of those of steel.

In the tests made on bars high in phosphorus, the results reported by Mr. Summers should be supplemented by their size and the distance between supports; otherwise they are worthless. In the instance of the high-grade irons, erroneously called semi-steel, which showed such differences in permeability and fracture, would not the temperature in casting account for most of such differences?

It seems that Mr. Summers's experience has been principally with ferro-silicons; but, as every founder knows, these are to be used with great caution in cupola work, as such mixtures lack uniformity, and, as a consequence, the results are far from certain. It is much better to make use of irons of nearly the composition required in the charge than to mix up scrap and ferro-silicon, although, theoretically, this should be just as good.

In the discussion of that part of Mr. Summers's paper which concerns the relation of silicon to graphite, the tables given, and the deductions made therefrom, fail to be of value, because the temperature-conditions are not stated. The anomaly of a pig-iron with 3.36 per cent. of silicon, yet having all its carbon in graphitic form, which is attributed to local conditions, would be better laid to heat and cooling-conditions in the making of this exceptionally peculiar iron. Indeed, Mr. Summers subse-

quently declares that temperature controls the effect of silicon in pig-iron, as well as in iron from the cupola, the amount of air used to burn the fuel in a given time producing effects which must not be neglected; and it may be added that the amount of fuel with which the melted iron comes in contact, while dripping through it, affects the total carbon also.

Inasmuch as the carbon-question in pig-irons is a comparatively simple one, any specifications which include this element must be a burden to the furnace-man; and, again, any specifications requiring a minimum of graphite in pig-iron must make an expert founder smile when he thinks of the fact that chilled car-wheels, when remelted and poured "dull," give perfectly gray castings.

On the whole, the paper of Mr. Summers is a curious one, and well worthy of study by founders who wish to note to what ideas the copious use of ferro-silicon in daily mixtures gives rise.

R. A. Hadfield, Sheffield, Eng. (communication to the Secretary): I have read with special interest the excellent and most interesting communication by Mr. Summers on "Modern Cupola Practice." The title alone does not do justice to this important paper, as in several instances it offers solutions for results obtained from cast-iron alloys which, to the best of the writer's knowledge, have not been satisfactorily explained elsewhere. As an example, I would cite the very interesting theory stated by Mr. Summers, with regard to Professor Ledebur's temper-carbon, as being carbon in a "transition-state towards graphite."

I am glad to note that Mr. Summers regards as tenable the idea of the existence of a carbide of manganese. The hypothesis of such a combination appears to me much more satisfactory than the invoking of allotropy. It cannot be doubted now that carbide of iron exists; and there should, therefore, be little doubt as to the existence of another carbide in the same group of elements as that which includes both iron and manganese.

I am specially interested in the observations regarding the existence of silico-carbide, having myself produced some very peculiar alloys of silicon-iron containing carbon. They possess

special hardness—in fact, when properly treated, almost that of self-hardening steel, without the brittleness of the latter. Although silicon has been ordinarily a despised element, its use, when properly applied, is much more serviceable than is ordinarily imagined.

On the other hand, as Mr. Summers properly states, in ordinary foundry-practice this metalloid has often had attributed to it qualities, both good and bad, for which the real responsibility was due to other causes, some of which he himself has well and clearly shown. This has arisen, no doubt, from the fact that the influence of silicon on carbon is very important, and that the results observed were due rather to the condition in which this element was found than to the amount of silicon *per se*.

In conclusion, I would again say with how much interest I have read this paper, feeling sure that, to the practical man, who often sees that current theoretical explanations do not always fit in with the facts he notices, it will specially appeal. As a British member of the American Institute of Mining Engineers, I hope that we may have more communications from Mr. Summers, and that he may see his way to put his views before our Iron and Steel Institute for discussion on this side.

Mr. Summers: Replying to Mr. MacPherran, I would say that I cannot agree with him that total carbon varies too much to permit much dependence to be placed upon its determination. I have frequently checked this point, and find that, when care is exercised, the carbon-determinations are nearly as uniform as those of silicon. The drillings should be taken from different positions. Six pigs compose our sample, equal parts of each being taken. Mr. MacPherran has mistaken my meaning in this regard. I do not claim that one can judge of the softness of pig-iron from the total carbon, but that one can certainly do so from the graphitic carbon. In the specification to which he refers, a graphitic-carbon limit is also specified, which puts another phase on the question. I believe firmly, and this is confirmed by practice, that the carbon-specification is more important than that of silicon.

Without attempting to account for the variation of 0.7 per cent. in silicon mentioned by Mr. MacPherran, I must say

that, if his point is well taken, it seems to me we should look for considerable effect upon the graphitic carbon. I am unaware that any positive claim is made in my paper that the blast could increase the silicon to the degree mentioned. Mr. MacPherran seems to have better control of his burden than is common in normal practice.

As to the remainder of Mr. MacPherran's remarks, they all state admitted facts of foundry-practice, and I cannot see that they affect the data given by me. As to the size of our castings, I will say that they were all taken from the same pattern, 0.5 by 0.5 by 12 inches in size, giving a bar too small to permit any noticeable amount of segregation. Moreover, as great care was taken to maintain the same conditions for each casting, it is doubtful if any anomalies can be traced to different rates of cooling, etc. This conclusion is confirmed by the fact that, with the same charge, I have found the graphite the same in castings made from this pattern at different times.

Dr. Moldenke's remarks are so based on orthodox foundry ideas and opinions that I can scarcely make any reply to them. He is mistaken, however, in understanding that I have labored to "present a theory at variance with the conclusions of all the leading scientific founders." I do not claim originality for the "theory," as he calls it; but I have found that it explained anomalies in foundry-practice which I could not explain by the ideas in which I had been schooled.

Dr. Moldenke asks how one can govern the graphite if the silicon be left out? If he can govern the graphite with the silicon, he is more successful than I have been. I do not attempt to govern the graphite. I simply get as much graphite as I can, and control the total carbon by mixing high- or low-carbon iron or scrap, as the case may demand. Again, if we want fine machinery-iron, I think it is accepted as the best practice to fix the silicon at about 2 per cent. How, then, if we thus fix the silicon, and if it controls the graphite under all circumstances, are we to control the graphite? Perhaps Dr. Moldenke would not fix the silicon at any fixed amount, but would vary it until he got the graphite-content he wished. If so, I can only say that he differs from the best practice. The two most widely known and recognized manufacturers of fine machinery do not follow such a plan. The analyses of their castings show:

	I. Per cent.	II. Per cent.
Silicon,	1.90	1.87
Sulphur,	0.092	0.116
Phosphorus,	0.822	0.656
Manganese,	0.325	0.325
Graphitic carbon,	3.29	3.15
Combined carbon,	0.17	0.51

These castings were small, and one was a standard test-bar, 1 by 1 by 12 inches in size. Evidently, in this case, silicon has not been the controlling element, and has been kept as low as one well could keep it in foundry-practice for this grade of work. To what, then, was the percentage of graphite due? If the conditions mentioned by Mr. MacPherran and Dr. Moldenke are responsible for these results, then conditions are more potent than silicon in producing graphite, which is the point I wished to bring out in this connection, though I have not endeavored to define the conditions. I think Dr. Moldenke will find that a larger number of scientific founders differ from him than he is aware of.

Regarding the statement that the limit 0.10 per cent. for sulphur is too high, I can only refer Dr. Moldenke to the above analysis, which represents one of the best machinery cast-irons on the American market; and also observe that, in normal foundry-practice, sulphur is seldom below 0.06 per cent.

As long as "semi-steel" is an accepted term in the trade, I can see no objection to its use, especially as the semi-steel referred to is about 50 per cent. steel. The name is not new or original with me, and, as far as I know, does not purport to describe the properties of the metal.

In my opinion, the use of ferro-silicons is too general in this country to need discussion. There are scarcely any foundries where they are not used. I do think, however, that they are often employed with too little discrimination.

Dr. Moldenke's "expert founder," who does the smiling at graphite-specifications "when he thinks of the fact that chilled car-wheels, when remelted and poured 'dull,' give perfectly gray castings," can probably explain why it is difficult to get a soft gray iron to chill, although the silicon is low. In this connection one may be excused for thinking that an iron of high graphitic content, though it may be low in silicon, is frequently

of good use as a softener, since experience seems to show that it is easier, by the use of it, to make a casting high in graphitic carbon under normal conditions. I confess, however, that this proposition is open to dispute.

I would say that I am not an advocate of "the copious use of ferro-silicon." In fact, it is against this practice that my paper is directed. But I am far from agreeing with Dr. Moldenke, if he means to say that their use should be avoided.

With Mr. Hadfield's ideas regarding manganese I am in accord. It is by no means certain that this element cannot be used with advantage in foundry-practice to prevent oxidation, etc. The marked effect of manganese on both the permeability, hysteresis and permanent magnetism of iron and steel certainly points toward the existence of a carbide of manganese.

My paper was written, not to advance a new theory, but to invite discussion, and to discover whether others, as well as myself, have found difficulty in controlling foundry-mixtures by means of silicon. It does not seem sensible to talk of controlling cooling-conditions in commercial practice; and if foundry-conditions are more potent than silicon in controlling graphite, some, at least, of our current beliefs and traditions should be modified.

Since writing the above reply to the criticisms of my paper, I have received and read with much interest Mr. Bachman's paper. His tabulated data are so complete that I can add scarcely anything which would give weight to the argument. They are, indeed, so much superior to mine that I cannot help but feel that the points contended for are, to say the least, based on some foundation of fact. His numerous analyses confirm my own experience in every way; and many more such analyses as are presented in his paper could be produced, in which one would be unable to trace any appreciable relation between the content of silicon and that of graphitic carbon.

I wish to record my support of Mr. Bachman's criticism of the work done in support of what I may call the "silicon theory." It is well known that Mr. Keep's work was far from complete from a chemical standpoint, as well as from a metallurgical one. There were so many conditions in Mr. Keep's work that might have accounted for his results as to permit

serious doubts concerning his deductions. I myself have prepared irons by melting in graphite pots mixtures of ferro-silicon iron and wrought-iron drillings, which certainly make very curious alloys.

The following are the analyses of some of these pot-irons:

	I. Per Cent.	II. Per Cent.	III. Per Cent.	IV. Per Cent.	V. Per Cent.	VI. Per Cent.	VII. Per Cent.	VIII. Per Cent.	IX. Per Cent.
Silicon,	1.14	0.58	0.916	1.037	0.504	0.134	1.776	2.546	2.60
Manganese,	0.228	0.062	0.359	0.227	0.132	0.425	0.661	0.698	0.49
Sulphur,	0.046	0.038	0.059	0.055	0.048	0.057	0.074	0.052	0.04
Phosphorus,	0.412	0.269	0.723	0.794	1.049	0.664	0.886	1.139	1.15
Gr. carbon,	0.006	Trace	Trace	Trace	Trace	Trace	Trace	Trace	None
Com. carbon,	2.006	2.519	2.63	2.51	2.59	1.27	1.77	1.74	1.15

These irons were compounded for the purpose of studying the effect of the different elements upon the magnetic permeability of iron; and a more complete description of them can be found in the *Journal of the Society of Chemical Industry* for December, 1897.

In no case was the graphite over 0.06 per cent., and, the percentage being so small, the determination was neglected. The irons, some of which were covered with charcoal in melting, absorbed large contents of carbon, nearly all of which remained in the casting as combined carbon.

It will be noted that the silicon varied from 0.5 to 2.6 per cent. This has not affected the peculiar composition of the iron in any way. I cite these results to show the errors liable to arise in deductions made from small experimental heats and in basing foundry-practice upon them. Especially is this true where accurate chemical analysis does not accompany the work.

It seems to me that the advocates of the effect of silicon upon carbon are a little too bigoted in their advocacy of the theory, and frequently a fair hearing is not given to data presented in opposition. I certainly think the matter worthy of further investigation, every care being taken to discover the truth; and it does not seem to me advisable that these contentions should be scorned simply because they are not concordant with established belief or individual opinion. In support of this view, the valuable data presented by Mr. Bachman should have great weight.

The Use of the Tri-Axial Diagram and Triangular Pyramid for Graphical Illustration.

Discussion of the Paper of Prof. H. M. Howe, presented at the Atlantic City Meeting, February, 1898. (See p. 346.)

R. H. THURSTON, Cornell University, Ithaca, N. Y. (communication to the Secretary): The "tri-axial diagram" was, I think, first employed by me in the work of the "U. S. Board Appointed to Test Iron, Steel and Other Metals." A model was exhibited to the American Association for the Advancement of Science, and the diagram was first published, and the model described, in a paper presented to the same society at its Nashville meeting in 1877.* The problem then presented to its inventor was to find a method of planning a research involving proportions of ternary compositions in such manner that, a limited number of such compositions being tested, the results should give the law of variation in the physical properties of all possible combinations of the three elements, and the actual numerical values of such as were important to the investigator. The investigation of the binary alloys, copper and tin, copper and zinc, and tin and zinc, had been completed and satisfactorily discussed, employing the usual bi-axial diagram, of which the ordinates represented the one, and the abscissæ the other, of the characteristic properties to be studied, the composition being commonly indicated by the abscissæ. In these cases it was easy to ascertain, by the study of a limited number of compounds, the law of variation in strength, ductility, etc., with composition; but, for a long time, the ternary alloys were not attacked, because no simple and satisfactory method presented itself of securing results significant for the whole field by the investigation of a comparatively small number of alloys.

The tri-axial diagram was finally hit upon as a means of representing and interpreting results, and researches were at once begun, and in due time satisfactorily completed.

* See *Proceedings A. A. A. S.*, 1877.

This device rendered possible the doing of the work at comparatively small cost with a small number of specimen-alloys, yet so as to indicate with certainty the law of variation of quality with composition over the whole field, including every individual composition of the infinite number possible.* It permitted the identification of "the strongest of the bronzes," and exhibited the limitations, in practical use, of this class of compositions. The model constructed as the glyptic representative of the graphic construction is pictured in Fig. 1, and the corresponding contour-map is shown in Fig. 2.†

Perhaps the most interesting outcome of this investigation was the revelation of the extremely limited area of the useful compositions on the diagram. In the "copper-corner" there are to be found practically all the useful alloys for constructive purposes, and these only on the bronze side, within the limit of 20 per cent. of tin, while on the brass side the limit was about twice as large a proportion of zinc. The "maximum alloys" were found to cut a peak, shown on the model, Fig. 1, very clearly, at about the point of copper, 55; zinc, 43; and antimony, 2 per cent. Tobin's alloy (Cu, 58.22; Sb, 2.30; Zn, 39.48) represents chemical proportions, and is close to the peak. The rapid loss of strength, and the accompanying sudden decrease in ductility, when an excess of tin or zinc is employed, is seen in the rapid fall, shown by the model, from the upper peak and its adjacent plateau to the lower levels. Since the original research, repeated investigations have identified the "maximum alloy" very accurately. Pure copper and zinc and "phosphor-tin" were used in the most accurate work.‡

The contour-map, Fig. 2, constructed from such a model, or from the original data, constitutes the best quantitative illustration for the purposes of the engineer, and, perhaps even better than the model, exhibits the series of compositions which offer similar qualities. Either model or diagram can evidently

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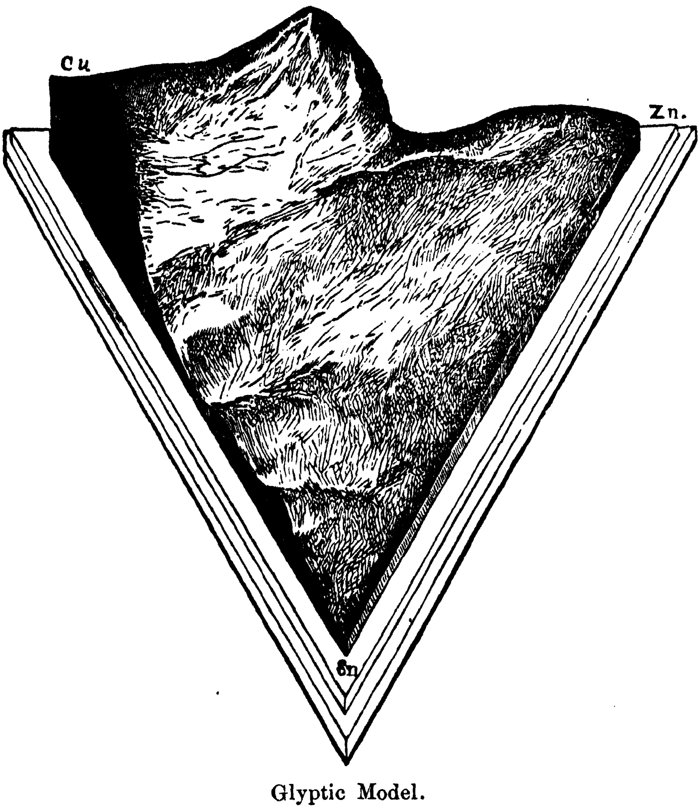
* This saved, probably, several thousand dollars, or a wasted research.

† Figures 1 and 2 are taken, by permission of the publishers, John Wiley & Sons, New York, from Thurston's *Materials of Engineering*, vol. iii., pp. 427 and 435.

‡ See Reports of the U. S. Board Appointed to Test Iron, Steel and Other Metals: Washington, Government print, 1878, 2 vols., 8vo.; also Thurston's *Materials of Engineering*, vol. iii., chap. xi.

be constructed to represent any designated quality, such as tenacity, ductility or resilience, specific gravity or specific heat or conductivity.* The general idea of such graphic and glyptic representations of series of data, illustrating physical laws, is as old as Aristotle; but these particular forms are, I think, comparatively novel, and were probably first used by the writer in the connection above indicated.†

FIG. 1.



Glyptic Model.

Aristotle's diagram, illustrating the case of the exchange between shoemaker and builder, is perhaps the earliest recorded attempt at diagramming a scientific problem.‡ Watt and Clapeyron represented graphically the laws of the variation of pres-

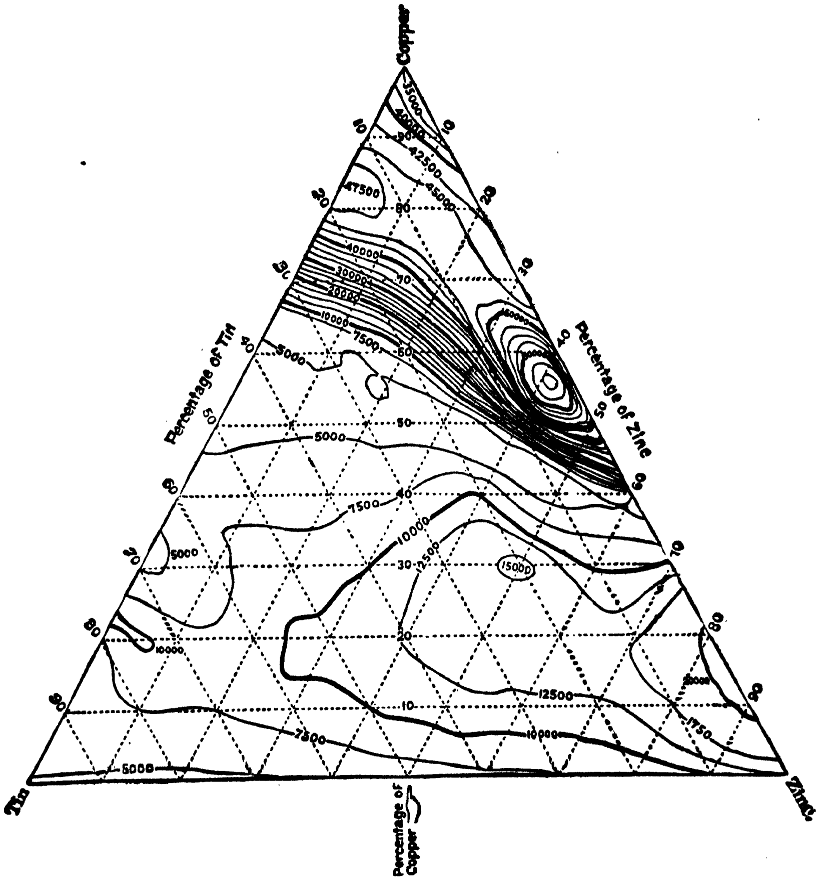
* *Ibidem;* also Thurston's *Materials of Engineering,* Part III., page 435.

† See *Trans. A. S. Mech. E.,* 1898, vol. xix., "Graphic Diagrams and Glyptic Models," by R. H. Thurston.

‡ Aristotle's *Ethic,* chap. vi.

sure with motion in exhibiting the action of the steam in the steam-engine cylinder,* and Gen. Morin employed the now familiar "stress-strain" diagram in discussion of the properties of the materials of construction, as illustrated in their behavior in the testing-machine. Since his time such systems of repre-

FIG. 2.



Contour-Map Corresponding to Fig. 1.

sentation have come to be "familiar as household words" in professional work, and in all scientific literature.

The graphic or the glyptic system of illustration may be made to accomplish directly, simply and accurately, certain results, never readily attained through ordinary tabulations of data.

* See Thurston's translation of Carnot's *Motive Power and Heat.*

A curve in a diagram, or a curved surface on the model, may be fully established in its every point by the determination of a comparatively small number of data. Five, or seven, or nine points settle the locus of a curve and its law; a few scattered points determine a surface; and the curve or the surface, thus determined, gives the position of the infinite number of points in either, any one of which may have, for the engineer, physicist or chemist, peculiar interest and value. A maximum or a minimum may be thus readily identified, although the experiments may not have touched that point, or in any way shown that such a maximum or minimum exists. Thus the composition of the "maximum alloy" among the "kalchoids" is found, or the proper distribution of steam in the engine is determined, in such manner as to lead to that arrangement and that method of operation which will insure, so far as this element affects them, the payment of maximum dividends on any investment.* This last is the ultimate problem of the engineer, whatever the work in hand.

Professor Howe, in the paper under discussion, now presents to us still another novel and useful application of the tri-axial diagram and of its congeners, and their glyptic representatives. Any ternary data may be obviously illustrated by the system, when the total of the three elements is a constant, and the elation of quantity may be exhibited in percentages. This application is most unexpected, and opens a field for the use of the system, in the work of the chemist, in a most interesting and promising manner. This method of exhibiting the fusibility and the heat of solidification of slags gives not only a means of determining the law of general variation of the composition of these compounds, but permits the identification, where it exists, of the composition of maximum or of minimum value for certain purposes, and with absolute precision, as the result of a limited number of analyses. In fact, it thus exhibits the quantitative value of any one of the infinite number of compositions, in the whole area exhibited, as perfectly as if an infinite number of analyses had been made.

Professor Howe's *résumé* suggests the use of this system as a check upon the results of analysis. This is, to my mind, one

* *Id.*; also Thurston's *Manual of the Steam-Engine*, vol. i., chaps. vii. and viii.

of the most valuable characteristics of this whole class of graphic and glyptic illustrations, and I constantly use the curve, or the surface, thus formed, in the checking of research-work. Data falling precisely upon the curve constructed from the observations are obviously accurate; but, if falling outside the line, they are as evidently inaccurate, and the trend of the line gives a means of establishing their real values. In our work, on strength of materials and on the steam-engine, particularly, as well as in much of our electrical-engineering and marine-engineering work, we are constantly using this system of record for this latter purpose, quite as much as for any other reason. These graphic and glyptic methods are admirable as tests of accuracy and of consistency in experimental work, in all important researches in applied science, and especially in engineering. Their extension to a fourth-dimension system is sometimes desirable; and there can be but little doubt that many useful applications will be found both for Mr. Howe's new method and for the other forms described. The metallurgist and the chemist certainly owe the writer of the paper under discussion most hearty acknowledgments of their renewed indebtedness.

FRANK FIRMSTONE, Easton, Pa. (communication to the Secretary): Prof. Howe's paper gives a very interesting application of the tri-axial diagram, by which some of the results of Prof. Åkerman's researches are presented in a clear and practical form.

The proposed pyramid-diagram, whereby the method is extended to the case of four variables, promises to be very useful, and it seems worth while, therefore, to indicate that a point may be easily constructed from its co-ordinates by the use of only one projection of the pyramid, namely, that on the plane of the base. Fig. 3 illustrates this proposition. In this figure, OXY, being the base of the pyramid (the altitude of which, as proved by Prof. Howe, is equal to the altitude of the equilateral triangle constituting its base), Z' being the projection of the vertex on it, H being (as in Prof. Howe's paper) the altitude of the pyramid, and v the vertical co-ordinate of Y', we have for any point Y' on an edge:

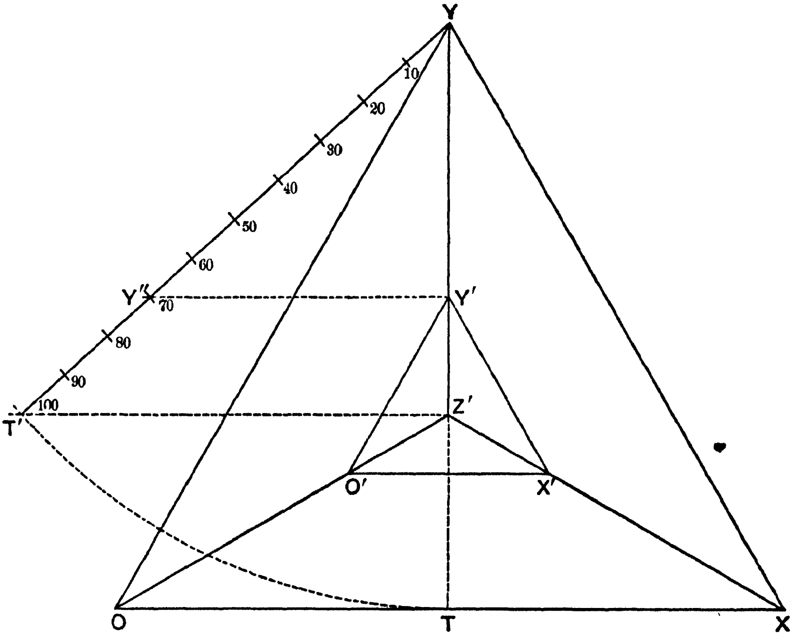
$$YZ' : YY' :: H : v;$$

$$\text{but } YZ' = \frac{2}{3} YT = \frac{2}{3} H; \text{ hence}$$

$$YY' = \frac{2}{3} v.$$

We can therefore get the value of v by measuring YY' with the same scale used for the base of the pyramid, and increasing the

FIG. 3.



Projection of Equilateral Pyramid on the Plane of Its Base.

value thus found by one-half. Or we can get v without calculation by drawing $T'Z'$ parallel to OX, and making $YT' = YT = H$, whereby any value of v laid off on YT' and projected on YZ' gives at once the intersection of the edge YZ' with a plane parallel to the base and at a distance v above it.

The isocals in the pyramid-diagram will not, in general, be plane curves, and will therefore differ more or less from their horizontal projections; but in many practical cases this could probably be neglected. In a series of silicates, having from 30 to 40 per cent. SiO_2, if v were taken for the value of the

silica, and the CaO, MgO and Al_2O_3 were laid out on the other axes, the isocals could not vary greatly from plane curves.

It is unfortunate that silicates of this series are not included in Ackerman's tables, for the tables and diagrams, as given in *Stahl und Eisen*,* show important saving in total heat of fusion by properly proportioning the magnesia to the lime in the case of the higher silicates with three bases; and there may be like differences among those more suitable for slags for furnaces using mineral fuel.

In practice, however, as Prof. Howe has well said, other considerations than a minimum heat of fusion will often, and perhaps generally, rule.

H. M. Howe, Columbia School of Mines, New York City, (communication to the Secretary): Since my paper was written, I learn from Professor H. LeChatelier, through Professor S. Jordan, that an American "savant," Gibbs, anticipated Professor Thurston in the use of the tri-axial diagram. On looking into the matter, I find on page 176, of volume iii., Transactions of the Connecticut Academy, that Professor J. Willard Gibbs in 1876 used the tri-axial diagram in the same general way as Professor Thurston. ("On the Equilibrium of Heterogeneous Substances," *op. cit.*, page 108.)

Professor Gibbs explicitly disclaims priority of that use, believing it more than probable that a like application had been made before him. (Private communication, April 22, 1899.)

That another distinguished man of science should have invented this diagram independently, of course does not in the least detract from the value of Professor Thurston's invention of it.

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* No. VI., 1886, *Blatt* xx., Figs. 4 and 5.

Tuyeres in the Iron Blast-Furnace.

Discussion of the Paper of Mr. Hartman. (See p. 666 ; also, for additional discussion of the subject, pp. 673 and 858.)

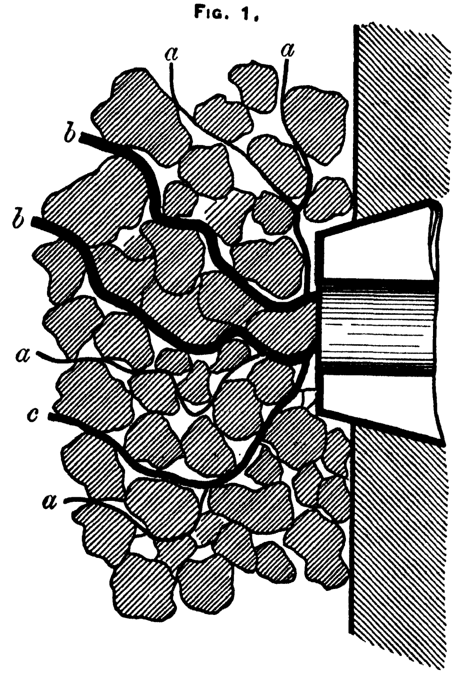
L. S. AUSTIN, Denver, Colo. (communication to the Secretary): Mr Hartman says (p. 4 of the pamphlet) that the penetration of air into the crucible of the blast-furnace "is, of course, a function of the diameter of the nozzle." To this I would emphatically take exception.

Fig. 1 represents a vertical ideal section of tuyere, crucible-walls and charge, immediately at the tuyere, in which the currents of air are represented, both in quantity and in direction, by the tortuous black-lines emanating from the nozzle-opening and passing through channels of the least resistance. As an electric current, flowing by various conductors to a common point, is carried in quantity inversely as the resistance of those conductors, so here the air ramifies through the most open cavities or channels, relatively little of it passing where the resistance due to fine materials is great. This figure represents the course of currents in the vertical plane only. As a matter of fact, they pass away in every plane, as opportunity offers. The effect of blocking or decreasing the area of the nozzle-opening would certainly only decrease the quantity of air entering by that particular tuyere, without assisting penetration. The large channels (*b*, *b*, Fig. 1, for example) would still carry the larger part of the air, while again the smaller channels (*a*, *a*, *a*) would continue small and take no larger percentage of the air than before. These channels momentarily change, however, with the slow descent of the coke. Further on (p. 5), Mr. Hartman describes the dispersive effect of a single lump of fuel. He has only to conceive the entire spaces before the tuyere-nozzles as filled with such lumps, in order to parallel the condition of affairs I have just explained.

It is where the large pieces of the charge come down, that the mass is the most open, and the blast passes most freely. On the other hand, wherever the fine ore collects combustion proceeds slowly, because of the smaller supply of air. The

metallurgist used to lead- and copper-smelting has been accustomed to look at the surface of his charge constantly, to watch the action of the blast as it rises at the walls and in the center, and to feed his coke and his charge so as to secure uniform running. If his charge settles irregularly, he at once knows the fact, and takes his measures accordingly. Therefore, to dump large quantities of materials into a furnace by means of a bell-and-hopper, without knowing exactly what becomes of it, seems to him crude practice. We know very well that a

FIG. 1.



Diagram, Indicating the Distribution of Blast from a Tuyere.

quantity of mixed large and small materials is apt to separate as it slides down, the large pieces going to one place, the fines to another. Were it possible to deliver and maintain charges in uniform distribution, the blast would rise uniformly at the top. If the fines are charged most largely at the sides, and the larger pieces at the center, then the blast will go quite freely to the center. In other words, penetration depends on the disposition of the charge; and, no matter how hard the furnace is blown, the air will pass away by the channels of the least re-

sistance. If the charge is put in right, it should come down right.

Mr. Hartman says (page 5), "The hotter the fuel in the crucible, the greater the avidity of air for the fuel, and the farther in it will go." While I admit that intensity of combustion increases with the temperature of the fuel, I do not admit that on that account, the air will penetrate farther. This effect of the combustion, indeed, is to increase the volume of the gases by the conversion of carbon into its gaseous compounds, the removal of which, however, still depends upon sufficient channels to convey them away.

A common way of looking at the action of air is to conceive it as acting like a jet of water, the momentum of which would carry it the farther, the greater its velocity of issue. Air, however, being an elastic fluid, expands at once into the cavities it enters; or, perhaps, we may say that these cavities contain air under pressure, expanding in every direction. A curious experiment illustrating this may be found in West's "Metallurgy of Cast-Iron," p. 190. The full blast was permitted to enter an empty cupola, 66 inches in diameter, and yet a handkerchief, held in front of and one foot away from the tuyeres, was scarcely disturbed by the blast.

For a further discussion of the subject, much on the same lines, see Percy's "Metallurgy of Iron and Steel," ed. 1864, pp. 481 to 487, inclusive, entitled, "Descent of Solid Materials" and "Ascent of Gaseous Current."

I think it follows, from an acceptance of the correctness of the views which I have set forth, that the form of the tuyere is a matter of indifference, and that it may be made round, oval or rectangular, without affecting in any way the distribution of the blast. It seems to me, also, that an increase in the number of tuyeres would be an advantage, since it breaks up the spaces otherwise existing between adjacent tuyeres, utilizes channels close to the walls, otherwise little used, and makes more vigorous combustion at those points. It was formerly considered important to point the tuyere-nozzle downward; and I know of two patented applications of this principle. That the idea is fallacious is, I think, now generally recognized; and in lead- and copper-practice tuyeres are set horizontal, that the slag may the more easily flow away by the slag-escape, in case it enters the tuyere.

R. W. RAYMOND, New York City: While the principles to which Mr. Austin appeals are undoubtedly operative, I think their effect may be masked by others quantitatively more potent. As I understand it, the proposition that in the iron blast-furnace the penetration of the blast is a function of the nozzle-diameter, refers to a horizontal penetration to the center of the hearth, and not to an upward distribution. In cupolas and small blast-furnaces, there may be practically no difficulty in getting air to the center; but in large blast-furnace hearths, 8, 9 and more feet in diameter, the very causes which Mr. Austin cites may absorb the energy and divert the direction of the blast until little (or perhaps even none) of it reaches the center. Under such circumstances, a decrease of hearth-diameter might be a remedy, but it would involve other considerations, which need not be discussed here. The pertinent question is, Would a change in nozzle-diameter (the quantity of air delivered being the same) produce any effect in this respect, and, if so, what effect? I specify the quantity of air because that is the measure of the smelting-capacity. Of course, the delivery of the same quantity of air through a smaller nozzle would mean a greater velocity and a higher pressure. These two elements constitute the energy of the blast-current; and I do not think Mr. Austin is quite correct in assuming that the increase of initial energy thus obtained would have no effect upon the distribution of the blast. Conceding that its progress upward might still be distributed among the channels he describes in inverse proportion to their resistance (*i.e.*, in some direct relation to their diameter), I still conceive that the increased initial energy might add to the horizontal section through which the gases ascend a central area not reached effectively by a smaller energy. This is, in fact, our experience with large hearths and high pressures in the metallurgy of iron.

The instance cited by Mr. Austin from West's "Metallurgy of Cast-Iron" is not conclusive, because it is not quantitative. It does not state, for instance, what was the force of the "full blast" allowed to enter the empty cupola. And Mr. Austin's inference from this experiment proves too much. For if it confirms his theory that air delivered in a jet expands at once into the cavities it enters, it ought to follow that a jet of air

delivered into the open atmosphere (where the pressure resisting its expansion would be less, and the said expansion, consequently, freer than in a closed space already under more than atmospheric pressure) would not move a handkerchief hung in front of it. To disprove this conclusion, one need only blow out a candle.

The fact is, that air, being elastic, does, as Mr. Austin says, expand when introduced into spaces of lower pressure; but it is not true that this expansion takes place "at once," in the sense of consuming no appreciable time, and of destroying instantly the momentum of the jet in the direction of its axis. The distribution by expansion goes on, in closed space, no doubt, until the pressure is equalized throughout; but during this process there are currents in motion, and one of these is the current of the jet itself, which will persist, though with diminishing energy, for a longer or shorter distance and time, according to the relation between its initial energy and the resistance it encounters. If Mr. West's cupola-blast did not stir a handkerchief one foot from the nozzle, a blast of greater pressure and velocity would have done so.

Perry's book, though a treasure of information as to the state of the art when it was written, is scarcely a safe guide in problems involving much larger production and higher blast-pressures than he considered. So far as I recall, it does not directly deal at all with the question here under consideration. On the other hand, I find that Ledebur, our best present authority, makes some remarks,* of which I here give a closely literal translation:

"With the blast-pressure, measurable by the manometer, increases the velocity, and with the latter the energy of the blast. By higher pressure the blast is therefore enabled to overcome more successfully the resistances which oppose its advance (*Vordringen*) between the pieces of fuel, and to advance further in the furnace. The particles of oxygen impinge with greater energy upon the pieces of carbon, and enter into more intimate contact with them; the combustion to carbonic oxide is hastened; the combustion is condensed into a smaller space. In this respect, by the way, blast under high pressure operates

* *Handbuch der Eisenhüttenkunde*, 1893, vol. ii., p. 314, under "Einfluss der Windpressung und Windmenge."

like a heated blast. The denser the fuel—*i.e.*, the more difficult to burn—the higher the pressure which is advantageous.

"But the stronger advance into the furnace effects also, within the furnace's sectional area, a more uniform combustion; this is prevented from taking place chiefly near the walls, and leaving the middle of the furnace unaffected thereby. It follows that the ores descending in the middle are better heated and reduced than under too low a pressure; the measure of the reduction through carbonic oxide is extended. According to explanations previously given, this may result in a saving of fuel.

"In general, therefore, the blast-pressure should increase with the diameter of the furnace at tuyere-level."

Prof. Ledebur goes on to explain the effects of too high a blast-pressure, and of a pressure raised without decreasing the nozzle-area of the tuyeres. The quantity of blast, he says, must be fixed mainly by the capacity of the furnace and the reducibility of the ores smelted, since it measures the product. A given quantity of blast having been determined as normal for the maximum product of desired quality, the blast-pressure is to be adjusted according to the diameter and height of the furnace, the density of the fuel, and the physical character of the charge. It is clear, from his discussion of the subject, that he considers penetration of blast to the center of the furnace to be a desirable and practicable thing, and that a means of securing it without changing the quantity of the blast is to blow the same quantity through smaller nozzles. On p. 387 of the same volume he discusses one other means, namely, the projection of the nozzles into the furnace, so as to bring them nearer the center. But as this is aside from the present question, I will not quote his remarks concerning it. What I have given above represents not only the view of this authority, but also, I feel confident, the actual experience of blast-furnace managers of the present day.

With regard to the material occupying the space before the tuyeres, Mr. Hartman, in a note addressed to me, says (speaking, no doubt, of a large iron blast-furnace in normal running): "No ore is found within 10 feet above the tuyeres. If it gets so far, it is melted as silicate of protoxide of iron, and goes out with the slag. Thorough reduction must precede smelting, or

this loss in the slag will occur. The ore plays no part in determining the course of the currents of air. It is the fuel alone which does that. A piece of fuel before the tuyeres is oxidized in a few seconds."

I do not understand Mr. Hartman as meaning to deny that the distribution of ore above the zone of fusion affects the regularity of its descent and the uniformity of its reduction, by affecting the currents of gas (not air) which pass upward through the stock; or that, under abnormal conditions, even air-currents may find their way up into the zones where they should not be present, and there effect disastrous premature combustion of fuel, semi-fusion of stock, etc.

The Mines of the Frontino and Bolivia Company, Colombia, S. A.

Discussion of the Paper of Mr. Cragoe (see p. 591).

(Buffalo Meeting, October, 1898.)

Frank Owen, El Perú, Venezuela (communication to the Secretary): Mr. Cragoe's accurate description of the rich and extensive mines of the Frontino and Bolivia Co. is of much interest to those acquainted with Colombian mining. While the labor in that part of Colombia is, of course, admittedly far inferior to that of white miners, it is, nevertheless, decidedly superior to that available in other parts of the republic, and in the northern mining regions of South America generally. The Antioqueños have been miners for several generations, and many of those employed at the Frontino Co.'s works have been brought up at the mines, which is a distinct advantage. The ground encountered in the surface-workings of that district being very treacherous, they are skillful timber-men, every miner being able to put in his own timber in tunnels, stopes, etc., so that, except for the main shafts, where sawn timber is used, special timber-men are not required. In the shallow workings it is the custom for a miner to come out of the mine into the thickly-wooded bush and select and cut the timber he requires, which he will then take into the mine and put in

place. That they are not inferior miners is shown by their being capable of overhand stoping, to which the West Indian negroes in the mines of Venezuelan Guyana are trained only with considerable difficulty—though the latter are paid in gold coin and not in depreciated paper. A miner's wages to-day in this district are $2.25 Venezuelan (equivalent to $1.80 U. S.), while ten years or so ago, in the palmy days of El Callao, they ran as high as $5.00 Venezuelan ($4.00 U. S.).

The rapid decay of timber in the mines is, I think, attributable not to inferior quality, but rather to the hot and damp atmosphere to which it is exposed. The hard-wood timbers of the northern part of South America are indeed noted, even in Europe, for their strength and durability—for instance, green-heart, purple-heart, locust-wood and bullet-wood, "balatá" (a kind of India rubber), etc. Green-heart timber from Demerara was used for the lock-gates on the Manchester ship-canal.*

Amalgamation by plates inside the battery would be, I believe, a great improvement at Frontino, such free gold as is met with in the ore being mostly coarse. I remember that in cleaning up the mortar-boxes it was no uncommon thing to come across small nuggets, weighing up to 1 dwt.

Transport in Antioquia is slow and expensive, as it is necessarily accomplished entirely on mule-back, owing to the mountainous nature of the country; and the roads are always bad, on account of the very damp climate. It is, however, far cheaper, and even in many cases more expeditious, than in this part of Venezuela, where the roads are much more level and the climate is dryer, so that bullock-wagons and mule-carts are in use. Freight from the port of Las Tablas on the Orinoco to El Callao now averages about 3 cents (U. S.), and was formerly as high as 12 and even 15 cents per pound. Mr. Cragoe does not mention that heavy parts of machinery, such as cam-shafts, etc., are carried to the mines in Antioquia by a class of men, specially trained to that kind of work, who are certainly not highly paid, considering what they have to perform. I have seen men in gangs of thirty or so at a time carrying a piano over the hills to Medellin!

It is indeed surprising that hitherto so little attention has

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* *Proc. Inst. C. E.*, vol. cxxxi.

been paid in Europe to the development of the undeniably great mineral resources of Colombia. General Marceliano Velez, a Colombian politician of some note, and at one time a candidate for the presidency, has written of the "forests absolutely unexplored, filled with unheard of wealth, but left in isolation and solitude as complete as if they were shut off from the world, like the regions of Central Africa." M. Moulle, a French mining engineer of repute, in a report published in Paris in 1887, says:

> "Antioquia is undoubtedly one of the countries in the world where gold-deposits of all kinds are to be met with in the greatest profusion. If, thus far, her immense wealth has not attracted the attention of Europe, this is principally due to her position in the center of Colombia, a country which, until the last few years, was, so to say, completely unknown to the European public. After traveling through Antioquia for many months in search of information, I believe I may say that, with some exceptions, the gold-beds have barely been worked superficially, and, from the point of view of modern mining industry, they may still be regarded as virgin deposits."

Mr. Geo. Jenner, at that time British Minister to Colombia, wrote in 1894:

> "Under the Spanish dominion the gold-fields of Colombia were admittedly the richest in the world, and down to the year 1848 they furnished fully one-third of the whole supply of American gold. It is well known that the Spaniards worked the mines in a most primitive manner, and that the gold they extracted was almost entirely derived from the surface of vein or alluvial mines, and from washing the sands of the rivers. . . . . If this be the case, it can scarcely be denied that with scientific prospecting—of which the Spaniards hardly knew anything—and with scientific appliances, the amount of gold that is still to be gathered from the Colombian soil must be infinitely greater than that which was dug up during the Spanish occupation. The one advantage the Spaniards possessed was cheap labor. For nearly two centuries they obtained gold and silver in return for the lives of Indians, from whom labor was exacted in so unmerciful a manner that, though none of the mines were worked by deep shafts, so great was the mortality amongst the Indians that whole districts were depopulated, and many mines abandoned on that account alone. . . . . Others were abandoned owing to the inroads of the natives, and many more because the Spanish miners were incapable of overcoming difficulties that would offer no serious obstacle to the mining engineers of the present day. Of the richest mines known to the Spaniards many, therefore, remain practically untouched; and it is by no means improbable that careful surveys would lead to the discovery of others of at least equal wealth."

Much useful and interesting information regarding mining in Colombia was given by Col. E. E. Britten, in 1893, in a little work on the resources of the country, published in connection with the Chicago Exposition, at which he represented that republic.

Slips and Explosions in the Blast-Furnace.

Discussion of the Paper of Mr. Richards. (See p. 604.)

(Buffalo Meeting, October, 1898.)

J. M. HARTMAN, Philadelphia, Pa.: Mr. Fackenthal can remember some queer things that occurred at Durham, Pa., Aug. 3, 1876, while he was superintendent. The furnace was working stiff, *i.e.*, blast-pressure 10 to 11 pounds, which was high pressure for those days. With this pressure there was some circulation through the stock.

At 9 P.M., flushed cinder. After flushing, air was thrown off for a moment to "bot up"; the keeper turned the lever and threw on air just as the scaffold dropped, which eased up the pressure to 4¼ pounds. The air-receiver is twice as large at Durham as at other places. The air from this receiver, added to the two seven-foot blowers, gave an extra large volume, which, rushing into the furnace, fired the finely-divided carbon, causing an explosion that threw part of the stock in the furnace out into the open air, threw up the bell-and-hopper with the tunnel-head plates, and gave off a large volume of carbon-dust. The bell-and-hopper going up first, the stock followed, and, striking the bell-and-hopper, was deflected in a shower over the cast-house roof, breaking the slates. Sitting at the corner of the engine-house at the time, the stock pelted me, when I ran around the corner to get rid of it. Remembering there were two men at the furnace-top, I shut down the engines, and, running to the top, found them sitting in the hoist-house enjoying the fire-works. The explanation of this explosion is simple. There is in the blast-furnace a bed of fuel extending up, say, 16 to 18 feet above the tuyeres; in this bed of fuel some stray pieces of limestone are found, and a few pieces of unreduced ore; but, generally speaking, it is all fuel. On this bed of fuel is the zone of fusion. In this zone the descending stock is liquefied, except the fuel; the iron, melting, drops out of the ore, and the gangue unites with lime, forming cinder. The iron falls in the form of shot, and, with the cinder, trickles

through the bed of fuel, which reduces any oxides that escape reduction in the zone of reduction. This zone of fusion is about 3 feet thick, is constantly melting away at the bottom and being replaced from above. The best illustration of this zone occurred forty years ago, when old Uncle Dan Morgan ran the Danville furnaces. They were about 14 feet bosh by 50 feet high. One day the workmen struck, and nothing could be done but get some laborers to tap off the iron. Before the strike ended, the furnace chilled and set, so Uncle Dan put the men to work, digging out below. By and by no more stock would come down, and on examination a ceiling was found across the bosh overhead. They then went to the top and hoisted out the stock until they came to a floor; breaking through this floor, they found the cavity below. The zone of fusion had set firmly, and formed a division across the furnace. There have been occasions when this zone of fusion has come under my notice, so as to corroborate the Morgan experience. In shoveling out a furnace that is stopped when in good running order, the immense volume of coal-dust or finely-divided carbon is always noticed. Resting on the zone of fusion is the zone of carbonization, formed by particles of fuel that have escaped oxidization at the tuyeres and by the dissociated carbon. As the ore disintegrates and is broken up by carbon-impregnation, it is caught in this zone and thoroughly reduced. The better this is done, the better is the grade of iron. As Mr. Bachman says, the carbon is finely divided and red-hot. When the furnace is driven slowly, the ore and carbon collect faster than the fuel is burned below. The disintegrated ore and fine carbon passing down from above increase in height and form a dam, resisting the air and raising the pressure. As this allows only a small volume of air to go through the furnace, a scaffold ensues, stopping the descent of all material. When this scaffold finally burns off, there is a slip of the stock, which eases up the pressure and lets a large volume of air enter the furnace suddenly; the free air, rushing upwards, reaches the zone of carbonization, fires the carbon and explodes it. Generally speaking, all the scaffold seldom gives way at once; only part of it at a time; but when it does give way entirely, and a heavy pressure is blown, we have tremendous explosions.

The Durham furnace, in 1876, had a 7-ft. 6-in. crucible, with thin walls; 14,634 cub. ft. stock-capacity; bosh, 19 ft. 6 in., and stack 76 feet high. Owing to her limited tuyere-area, she could not use more than 10,500 feet of air. On comparing this with the stock-capacity, the ores reduced and disintegrated faster than the 10,500 feet of air could burn away the fuel; hence the high pressures of blast. After Mr. Fackenthal had increased the hearth in successive blasts to 11 feet and blown more air, the fuel was burned as fast as the ore was reduced and disintegrated. Excellent work was then done, and the product was more than doubled. This finely-divided carbon fired at the Philadelphia furnace, burning five men to death; and at the Paxton furnace the carbon settled on the man, fired and stuck to the skin until it burned off.

EDGAR S. COOK, Pottstown, Pa.: Irregular settling of the stock seems to be more or less frequent in every furnace that is kept up to a maximum product. The total output of iron will, of course, vary with the height of the furnace, diameter of crucible, richness of the ore-mixture, and quality and character of fuel used, whether coke or anthracite or a mixture of the two; but whatever the conditions may be as to size, etc., with a large continuous product for any particular furnace, slips and jumps seem to be inseparable from its working. This, at least, has been my experience, without much regard to the ores used. I have no doubt that certain ore-mixtures will add to this difficulty, or that the mixing of easily reducible ores with others that are very refractory, or the use of ores with a gangue that becomes pasty at comparatively low temperatures, will cause a furnace to hang and jump to a greater extent than under more favorable conditions.

I have not found that the use of 25 per cent. or 33⅓ per cent. of Mesabi ores, as compared with the same percentages of soft lake-ores from the Marquette and Menominee districts, has made any particular difference as regards irregular settling.

Several years ago we used 33⅓ per cent. of Mesabi ore, in connection with hard Lake Superior and a small percentage of magnetic, substituting Mesabi for a larger percentage of the latter in the hopes that this mixture would decrease the tendency to hang and jump. There was probably less disposition

to stop settling altogether, but in its place we were annoyed with little slips at frequent intervals. The stock on the top of the furnace moved but a few inches or scarcely at all. The pressure, as shown by blast-gauge, would vary but little. The draught from stacks of boilers and stoves would be discolored, but the volume of gas and the melting of the furnace would not be materially disturbed.

Occasionally these little slips would become dangerous. If they followed one another closely, say every five, ten or fifteen minutes, accompanied by an unusual amount of flue-dirt in the gases, we found that the series of slips would generally end with a "big jump," tightening the stock to such an extent as to run up the blast-pressure several pounds. Shortage of gas would follow, melting would practically cease, and the crucible would become cold.

While using the Mesabi ores we had plenty of jumps, but never any explosions of sufficient force to threaten damage.

For the last two years we have been using chiefly soft Marquette and Menominee ores in place of Mesabi. While these ores are to be preferred in some respects, as compared with the ores from the various Mesabi mines, yet, in so far as slipping and jumping are concerned, I am unable to distinguish any difference. It is about six of one to half a dozen of the other. The change was made with the view of overcoming some of the difficulties generally attributed to the use of Mesabi ores, but the results expected have not been realized in practice. There was no improvement at all in respect to more uniform settling. At present we are using about 16½ per cent. of Mesabi and 16½ per cent. of soft Marquette ore.

It is not to be supposed that serious irregular working, with slips and jumps, is the daily condition of any furnace. There is enough of it to prevent one from becoming stupidly lazy and dull, thus giving variety to the routine of furnace-work, but not sufficient to materially interfere with the practical results obtained; only we have constantly before us how much better the results would be if we could eliminate the irregular settling altogether, or find some method of controlling more absolutely the mechanical conditions prevailing in the interior of the furnace.

My interest in multiple tuyeres has been chiefly due to the

influence they might be found to exert upon the regular and uniform settling of the stock. So far as our furnace is concerned, I can drive it as fast as we can fill it, so that the exposure of ores to gases is not more than ten to twelve hours. This is somewhere near the economical limit, so far as our ore-mixture is concerned.

Under these circumstances I certainly would not adopt multiple tuyeres, with the extra expense and care their use involves, if the only benefit is to permit faster driving.

If, however, multiple tuyeres would aid in removing the tendency to slip and jump, they would be an untold blessing to the furnace-manager. So far as I am able to learn from several of my friends, who are trying the experiment, doubling the number of tuyeres does not seem to have any effect in that particular.

My information may not be sufficiently extensive; but so far as it goes, the irregularities of blast-furnace operation are just as great with the multiple tuyeres as with the ordinary number of tuyeres, fairly well-proportioned to the diameter of the crucible and the volume of air blown. The published report of the Duquesne furnaces for the month of October would seem to indicate that the ten-tuyere furnaces were doing practically the same work as the twenty-tuyere furnaces, the records being:

No. 1, 18,672 tons.
No. 2, 17,717 tons.
No. 3, 18,809 tons.
No. 4, 18,060 tons.

Furnaces Nos. 1 and 2 have ten tuyeres each, while Nos. 3 and 4 are equipped with twenty each. I cannot see the advantage of doubling the number of tuyeres, unless it will reduce the consumption of fuel and remove the tendency to scaffold and slip, which removal would be followed by a larger product of iron.

Slips and jumps are largely due to mechanical causes, related in a measure to tuyeres and character of ores and fuel employed and fluxing, but also, in part, independent of them all.

The walls of a furnace will keep perfectly clear and clean at times, and the furnace will run along with the regularity of clock-work, becoming almost monotonous. Just as the manager is about congratulating himself that he has at last solved

the problem of normal regular working, and has discovered the panacea for all furnace-ills, he is rudely awakened from his dream to find that serious trouble is threatening. Following a week of unusually gratifying results and a cast of exceptional tonnage, the furnace suddenly refuses to take any blast; pressure at tuyeres is abnormal; engine is barely able to run against the increased resistance of the stock to the passage of the blast; gas fails; steam falls off; and things generally have a very chilly appearance. The manager realizes then that certain indications have been overlooked or neglected in his fancied security, and that now the penalty is to be paid for presuming to become just a little conceited. In spite of all his efforts, improved constructions, more powerful engines, highly-heated blast, ore-mixtures, tuyeres, etc., finely-divided stock has gradually accumulated upon the walls of the furnace, either on the boshes or about the lower inwall; and not until the walls are comparatively clean and free from accumulations will there be a restoration of normal, economical work and consequent peace of mind. A repetition of this experience leads the manager to look for indications that will warn him of serious impending dangers; and, though he may not prevent entirely the disturbances he dreads, he can foresee and provide for them, so that one week of irregular working comes to be followed by others of good regular working, to be succeeded, sooner or later, by new irregularities. I guess this is the experience of most managers, and especially of those who come into close personal contact with the daily routine of furnace-practice.

Several years ago, we had a practical illustration of the locality of the zone of fusion, in a manner somewhat different from that of which Mr. Hartman has spoken. Our stock had been slipping more or less; but, as the blast-pressure was unaffected and the slag was hot and was melting in the usual quantity, little or no attention was paid to the succession of little slips, dependence being placed upon our new fire-brick stoves and the fact that we were using a larger percentage of coke than usual, instead of anthracite.

We paid the usual penalty of over-confidence, if not of ignorance. The small slips were succeeded by heavy jumps, and, before we could realize the gravity of the situation, the furnace ceased melting. The temperature of the crucible had fallen

to such an extent that the stock was becoming pasty around the tuyeres, threatening to close them with semi-fused iron and slag.

Orders were given to stop the engine and remove every alternate tuyere. We then commenced to rabble out the stock, wheeling away many cart-loads. After a time the men were driven away from the tuyeres by an outpouring of red-hot material, which ran with the fluidity of molten iron or slag. This proved to be flue-dirt, consisting of about 75 per cent. finely-divided ore, the balance being carbon (coal and coke), with a little limestone.

Rabbling was continued after the flue-dirt had been removed. At some of the tuyere-openings the stock was found to be hot and in good condition; at others, quite cold. Some of the anthracite coal was apparently untouched, having the same bright fracture as when mined; but the coke was all more or less incandescent. The limestone slacked on exposure to air and water.

Finally I was notified that the furnace, at the tuyeres, was empty. The furnace had been nearly full at the time of stoppage; so it was evident that there had been no settling of the contents. This was soon proved by the use of test-rods. Dynamite cartridges were exploded above the stock and in the cavity below, but without any effect. At last, my assistant and myself got up sufficient courage to climb into the tuyere-openings far enough to obtain a complete view of the interior. The bosh-walls were smooth and even, shining with the brilliancy of a highly-polished mirror.

Far above we saw an arch or dome, apparently some distance above the bosh. It was located more accurately afterwards by drilling holes. If the arch had given way while we were making these observations, I would not be here now to tell the story.

By the way, all of our troubles had been attributed by our men to the use of bronze bosh-plates, which had been adopted in place of bosh-coils. I called the attention of my assistant and the keeper to the fact that the skew-back of the arch was fully 10 feet above the top-course of plates, so that it was not at all probable that the plates had anything to do with the formation. Personally, I was satisfied of this from the study of

previous experiences, especially from 1880 to 1885; but this practical demonstration was worth more than any possible amount of argument.

The arch not giving way, we were in somewhat of a quandary as to the best course to pursue. Finally, I ordered the tuyeres to be replaced and the engine started, and the blast put on the furnace. The long stoppage, during which time cold air had been drawing into the furnace, had caused contraction to take place, so that the blast was able to pass through the arch, which, previously, it had been unable to penetrate. Upon starting up, the pressure was low, and gas came over freely to stoves and boilers; but as soon as melting commenced the pressure became high, and the flow of gas almost ceased. I was afraid we were going to have further trouble. It was about midnight—the worst troubles invariably occur at or about midnight. I was standing at the blast-gauge, watching the fluctuations of the mercury-column. The mercury suddenly dropped, and at the same time I heard a weird, rumbling sound in the furnace, followed by a terrible roaring and hissing. There was apparently no explosion. The first impulse was to run; but, on looking through the open door of the cast-house, I saw a pyrotechnic display that I shall never forget. The darkness of the night was illumined by burning gas, issuing under heavy pressure from a half-dozen or more of explosion-doors on gas-flues, while showers of red-hot coke and coal were falling all around.

Fortunately the bell on top of the furnace was wide open, else it would have been thrown out of position.

The disturbance was due to the fact that the arch had given way, causing the stock to drop 25 feet or more, driving coal and coke out of the top of the furnace and down through the gas-flues to the ground-level; then up through the checker-work and chimneys of the hot-blast stoves, 60 feet high, and I don't know how many feet higher, into the air.

After this diversion the furnace commenced melting, and went along all right. We have not had a repetition of this experience, being satisfied with our investigations as to the location of the fusion-zone.

F. E. BACHMAN, Buffalo, N. Y.: I have advanced the theory that to a large extent the difficulties are mechanical. I don't

mean to say that these explosions are mechanical, but that the ejectment of the stock, throwing out of tops, etc., are mechanical.

I see no reason at present, although I have had almost no experience in the use of large quantities of Mesabi ore, to change my ideas on the subject. That these explosions should follow irregular working or slipping in the furnace, I think can be explained.

I have noticed that immediately following slips there is little or no settling of the stock on top, although there is no lessening of the volume of gas passing into the stoves and boilers.

Supposing that, after a slip of 3 or 4 feet, the filling continues, but for some distance from the top the stock fails to move: there would be a much better chance for the deposited carbon to clog interstices in the stock, and for the additional effect in the same direction of the fine ore in the charge, than if the stock were properly settling.

Under such conditions it is easily supposable that the openings would be entirely closed, or closed so tightly that the pressure below the obstruction would be raised to a point approaching the blast-pressure, or to a point high enough to force a passage or an enlargement of the small passages which may remain. When this occurs, gas from below under pressure carries with it large quantities of deposited carbon and fine ore; if the case is aggravated, coke, stone, bell, etc.

As soon as the red-hot deposited carbon meets the air there are perfect conditions for a dust explosion, which naturally follows, the same as in a coal-mine or flour-mill.

SECRETARY'S NOTE.—The discussion of Mr. Scott's paper on "The Evolution of Mine-Surveying Instruments" (p. 679) will be printed in Vol. XXIX.

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[NOTE.—In this Index the names of authors of papers and contributors to discussions are printed in SMALL CAPITALS, and the titles of papers in *italics*. Casual references, giving but little information, are usually indicated by bracketed page-numbers.]

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